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MAGNETIC PROPERTIES OF HEUSLER ALLOYS

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MAGNETIC PROPERTIES OF HEUSLER ALLOYS

I. Introduction*

The Heusler alloys, so named in honor of their discoverer, are composed of copper, manganese and aluminum, and are remarkable for the fact that, in certain proportions, they are ferromagnetic, although the component metals are not ferro-magnetic.

The following investigation of the properties of these alloys is a contribution of data on the ultimate nature of magnetism. If the permeability of a magnetic substance could be increased, and also the hysteresis losses be decreased, it would evidently be of the greatest importance to electro-technics; hence an investigation on the ultimate nature of magnetism has a commercial importance as well as a theoretical value.

The methods of thermal analysis and photo-micrography have been used in an attempt to find relations between the magnetic and the other physical properties of the alloys. These methods have been described in considerable detail, inasmuch as they will probably be of value in the study of the whole subject of alloys, and especially in the study of steels and brasses. Thermal analysis accompanying a chemical analysis gives an insight into the nature of an alloy not possible to the latter alone, and photo-micrography is an excellent method of studying the crystalline structure on which numbers of physical properties so largely depend.

II. Historical Review

In 1901, Dr. Heusler reported[1] that he had accidentally discovered that certain manganese alloys were magnetic. While turning a certain alloy in a lathe, he had noticed that the turnings adhered to the tool. This discovery led to a general investigation of the magnetic properties of manganese alloys.

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*The thanks of the writer are due to Dr. A. P. Carman for aid and facilities, and to Dr. Chas. T. Knipp for his constant advice and kind supervision of the work.

[1] Verh. d. deut. Phys. Gesel. v. 5, p. 219, 1903.

Later, it was found by Dr. Heusler, W. Stark and E. Haupt[1] that a copper-manganese alloy itself was not magnetic, but, when mixed with zinc, arsenic, antimony, bismuth, or boron, it was more or less magnetic; and when aluminum was added, a good ferromagnetic alloy was obtained. In general, the properties of any given piece varied radically with the previous heat treatment, but the most magnetic alloy was formed when the manganese and aluminum were in the proportion of their atomic weights, i. e., manganese, 25%, aluminum 12.5% and copper 62.5%.

From this fact and also from the values of the molecular susceptibilities of certain salts, as given by G. Wiedmann[2], Liebknecht and Wills[3], and others, Dr. Heusler considered the case analogous to a salt solution in which the combination of manganese and aluminum in atomic proportions was the salt, and the copper the solvent.

The discovery of these alloys did not attract much attention in England until 1904, when R. N. Hadfield exhibited a sample at the meeting of the British Association, and soon afterwards Flemming and Hadfield[4] published the results of their investigation. In their experiments, they had made two rings which gave B and H curves similar to those of cast iron, with a maximum value for the permeability of 28 to 30 for $B = 1500$. They considered the magnetic properties to be due to molecular grouping and not as characteristic of the elements in themselves.

Andrew Gray[5] made a further investigation with two rods which he tested at low fields only. His specimens were fairly magnetic and improved slightly upon heating to temperatures below 400°C., but quenching from that temperature almost destroyed their magnetism. Neither vigorous tapping nor immersion in liquid air had much effect. The critical temperature was found to be about 350°C. Several quenchings produced a change in structure and, when viewed under the microscope, showed large nodules imbedded in a granular matrix.

Ross[6] working at the University of Glasgow on some specimens containing 25% manganese, 12.5% aluminum, a trace of lead, and the remainder copper, found large variations due to different thermal treatments. Prolonged heating below 100° C. had little effect, but distinct improvement was shown upon heating to 160° C., and a gradual deterioration on heating further to 220° C. Testing while immersed in liquid air showed a temporary improvement which was lost on returning to room temperature. Mr. Ross quenched his specimens at 50° intervals between 400°C. and 750°C. and then tested them, (1) as quenched, (2) in liquid air, (3) back at room temperature. The effect of quenching shows clearly in the permeability which gradually decreased to 550°C., dropped to a minimum of about half value at 615° C., and then rose to approximately the original value at 650° C. and above.

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[1] Verh. d. deut. Phys. Gesel. v. 5, p. 224, 1903.

[2] Lehre von Elektrizitat v 3, p. 958.

[3] Ber. d. deut. chem. Gesel., v. 33. p. 448, 1900.

[4] Roy. Soc. London. Proc., ser. A, v. 76, p. 271, 1905.

[5] Roy. Soc. London, Proc., ser. A, v. 77, p. 256, 1906.

[6] Roy. Soc. Edinburgh, Proc., v. 27, p. 88. 1906-07.

The magnetostriction was carefully measured by Austin[1] and found to give, for his particular specimen, a value $\frac{dl}{l} = 11 \times 10^{-7}$ for a field of 400 c. g. s. units, which is about one-third that for cast iron. The curves showing the relation of the magnetostriction to the field strength had the same general form as the magnetization curves, though rising more slowly.

Guthe and Austin[2]. found expansion curves similar to the magnetization curves and a numerical value about one-half that for soft iron. In general, the softer samples gave the greater expansion.

McLennan[3] found the continuous shortening, especially in strong fields, observed by Austin[4], and marked effects due the heat treatment and the difference in lengths of time after casting. The maximum elongation was observed in specimens having the manganese and aluminum in the proportion of their atomic weights. The permeability of some rings was found to vary greatly with the quenching temperature and was much greater when they were quenched from near their melting point than from a red heat.

Guthe and Austin[5] also tested their samples for Young's modulus and found it very large, although no exact results could be determined on account of flaws in the material.

Bidwell has shown, for iron and nickel, that there is a change in the thermo-electric power proportional to the magnetic expansion corrected for mechanical stress, but Guthe and Austin found no such effect for the Heusler alloys, even for a temperature difference of 100° C. Mendenhall reported in a private letter to them that he could find no evidence of the Kerr effect.

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[1] Ber. d. deut. Ges., v. 6, p. 211, 1904.

[2] U. S. Bur., Stds., Bul. 2, p. 297, 1906.

[3] Phys. Rev., v. 24, p. 449, 1907.

[4] ibid.

[5] ibid.

Hill[1] tried heating to various temperatures and found the effect reversible up to about 375° C., but at 500° C., irreversible.

In a discussion of the theory, Hill suggests that manganese may really be a magnetic substance under the proper conditions, as a consideration of the atomic susceptibilities and molecular magnetism of certain substances would indicate.

Defining "atomic susceptibility" as the mean susceptibility of the space containing one gram atom of substance in 1000 cc., Meyer[2] gives the following arrangement of elements in the order of their atomic susceptibilities: Ho, Er, Gd, Mn, Fe, Sa, Co, Yt, Nd, Ni, Pr. Liebknecht and Wills[3] give the following list of magnetic susceptibilities for certain salts:

Substance	k
Copper nitrate	.00163
Ferric nitrate	.01352
Ferrous sulphate	.01272
Manganic nitrate	.01536
Manganous sulphate	.01514

Yager and Myer[4] give the following values for the molecular magnetism of certain elements:

Substance	$I_m \times 10^6$
Nickel	2×2.5 c.g.s.
Cobalt	4×2.5 c.g.s.
Iron	5×2.5 c.g.s.
Manganese	6×2.5 c.g.s.

McLennan and Wright[5] studied the susceptibilities of several salts of the metals, copper, manganese and aluminum, and found approximately the same values given above with the additional value of —0.00018 for $Al_2(SO_4)_3$ and +0.00002 for $Al\,(NO_3)_3$. When combined in the proportions of the Heusler alloys, the susceptibilities followed the simple additive law and changed but little with concentration.

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[1] Phys. Rev. v. 21, p. 335, 1905.

[2] Ann. d. Phys., v. 68, p. 325, 1899.

[3] " " " v. 1, p. 178, 1900.

[4] Wiener Ber., v. 106, pp. 504, 623, 1897; v. 107, p. 5, 1898.

[5] Phys. Rev., v. 24, p. 276, 1907.

Hill[1] suggests that there is a simple explanation of many of the phenomena of the Heusler alloys in the allotropic theory of Osmond[2]. Thus iron shows the allotropic forms:

Alpha—soft, magnetic and stable below 700–780°C.,
Beta–hard, non-magnetic and stable between 750°C. and 860°C.,
Gamma—soft, non-magnetic and stable above 860° C.

In passing from one form to another, there is a transformation point, e. g., the common recalescent point in iron.

Nickel has only two known allotropic forms:

Alpha—magnetic and stable below 340° to 360° C.,
Beta—non-magnetic and stable above 360° C.

The transformation point can be depressed by alloying with another metal and the amount of depression is proportional to the amount of the substance added. Thus it is possible that the form of manganese stable at ordinary temperatures is non-magnetic, but that the alloying produces another allotropic form that is magnetic.

Zahn and Schmidt[3] studied the Hall effect and allied phenomena, chiefly the thermo-magnetic effect of Nernst and Ettingshausen[4], for the Heusler alloys and their individual constituents. They found the following values for the coefficient R of the Hall effect and for the coefficient Q of the thermo-magnetic effect.

Substance	$R \times 10^5$	$Q \times 10^6$
Heusler alloy	+ 1300	— 500*
Manganese	— 93	— 15
Aluminum	— 40	+ 20
Copper	— 50	+ 90
Lead	+ 9	+ 5
Iron	+ 1080	+ 1050

It is evident that the values of R and Q are not additive for the

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[1] Phys. Rev. V. 24, p. 276, 1907.

[2] The Crystalization of Iron and Steel, Mellor, pp. 12–15.

[3] Verh. d. deut. Phys. Ges., v. 9, p. 98, 1907.

[4] Wied. Ann., v. 29, p. 343, 1886.
" " v. 31, p. 760, 1887.

*The values for the alloy are average and approximate. The iron is inserted for comparison.

Heusler alloys, and comparison with iron shows no certain relation of sign or magnitude. The temperature coefficients of R and Q were found to be —0.003 and —0.006, respectively. Four other allied effects were detected, but not with enough certainty for quantitative measurement.

Recently, Knowlton[1] has succeeded in making specimens of especially good and uniform mechanical properties. He did not find any simple relation between composition and magnetic properties, and suggests that the copper has a much larger influence than is usually attributed to it. He made a study of the structure by means of photo-micrographs. Three distinct types of crystals were evident: (1) bright ones, unaffected by the etching fluid (HCl and $FeCl_3$); (2) dark ones, deeply etched, and (3) yellowish ones of the nature of a matrix. Crystals of the first type were evidently the magnetic ones, as no specimen from which these were absent was magnetic, and the value of I could be estimated with a fair degree of approximation from the area of these crystals. He found, also, no evidence of recalescence, and that the critical point for one specimen was 0° C. and for the others about 200° C.

III. TEST PIECES

1. *Preparation.*—Considerable time was spent in preliminary experiments with small quantities of copper, manganese and aluminum in order to learn the best method of alloying and casting. The test pieces were finally prepared in the following manner.

The materials were all tested qualitatively for iron, cobalt and nickel and found quite free from these, although they showed some carbon. Each combination of the metals was weighed out in a given proportion so as to make a total of about two pounds, melted in a brass furnace at the University foundry, and cast in fine sand molds. The melting crucible was a new clay-graphite one. The manganese and copper were put in first, and when they were thoroughly fused the aluminum was put in, a flux added, the mixture stirred with a well seasoned hard wood stick, and then quickly poured into vertical molds. Two rods, each 45 cm. long and 1.1 cm. in diameter, were poured from each melt and allowed to cool slowly in the molds.

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[1] Phys. Rev., v. 30, p. 123, 1910.

The ends of the rods, which were enlarged in the casting, were cut off and the whole length dressed down on an emery wheel to the form of a very prolate ellipsoid. The specimens were tested magnetically as cast, and again after grinding, and the grinding was shown to have had no effect upon them. Twelve samples, about 35 cm. long and 1 cm. in diameter, were thus prepared. None of the specimens could be satisfactorily worked and those with the larger per cent of manganese and aluminum were especially hard and brittle.

The magnetometer method was chosen as being best adapted to all parts of the investigation. A diagram of the apparatus is shown in Fig. 1. The resistance of the magnetizing coil was so high that 110 volts gave only two amperes through it, so this source of current was used for most of the work, but when a very small current was desired, storage batteries could be cut in at the switch S_1. The current was led through a 0–5 ammeter, reading to hundredths, and through an extra compensating coil C_2 to a reversing switch S_2, then through two control resistances, R_1 and R_2, and by the switch S_3, either through the coil C_1 alone, or through it and the magnetizing coil, MC in series. The extra compensating coil C_2 was made necessary by a small permanent field due to the wiring and the various instruments, and its position was located by trial. By means of the switch S_3, the magnetizing coil could be cut out and the coil C_1 used as a calibration coil, without change of position. Control magnets were used, so that

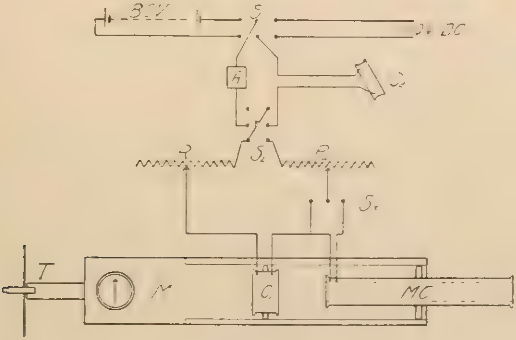

FIG. 1.

the field was about 0.275 c.g.s. The ammeter, switches, resistances, etc., were placed on a convenient table about two meters back of the magnetometer, while the magnetizing and compensating coils and the magnetometer were placed on a slideway about three meters long, clamped to a pier. The magnetizing coil was made one meter long with an average diameter of 9.2 cm., and was wound with two layers of No. 30 copper wire, on a copper water jacket with a water space of about 1 cm. For testing the specimens at higher temperatures, an electric furnace could be slipped inside the water jacket. The furnace consisted of a porcelain tube of 1.3 cm. inner diameter, on which was wound, bifilar, a nickel wire. The heating coil was covered with magnesite and placed inside a fire-clay tube which fitted closely inside the water jacket. The bifilar winding eliminated most of the field due to the heating current, and as soon as the coil reached about 350° C., the nickel lost its magnetic effect. For this arrangement of the magnetometer, the following formulas were used.

If

H_e is the earth's field as determined with the calibrating coil, then

$$H_e = \frac{2\pi n a^2 I}{10 r^3} \cot \psi \; \ldots\ldots\ldots\ldots\ldots\ldots \quad (1)$$

where n = the number of turns, a = the radius of the coil, d = the distance of the coil from the magnetometer, $r = \sqrt{(d^2 + a^2)}$, D = the distance of the scale from the mirror, $\cot \psi = \frac{2\mathrm{D}}{\mathrm{def.}}$ In the magnetizing coil

$$H = \frac{4\pi n I}{10 l} \; \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (2)$$

where n = the total number of turns, l = the length of the coil, and I = the current in amperes.

On the assumption that the magnetic length is two-thirds the actual length,

$$I = \frac{3\left(r^2 - \frac{l^2}{9}\right)^2 H_e \tan \phi}{4\pi a^2 l r} \; \ldots\ldots\ldots\ldots \quad (3)$$

where I = the intensity of magnetization, l = the length of the specimen, $2a$ = the diameter of the specimen, r = the distance from the center of the specimen to the magnetometer, and

$\tan \phi = \frac{\mathrm{def.}}{2\,\mathrm{D}}$

2. *Composition.*—The specimens were made up with the idea of testing some important points that had been developed in previous investigations; e. g., the effect of different percentages of manganese and aluminum, the effect of a little lead on the mechanical properties, and also the effect of a small amount of iron. Because there was considerable slag left in the crucible, it was believed that the percentage composition had changed somewhat in the process of melting, and so careful analyses of six specimens were made and the results are shown in Table 1.

TABLE 1

No.	Length cm.	Diam. cm.	Cu per cent	Mn per cent	Al per cent	Pb per cent	Fe per cent	IM* per cent
1	43.7	1.1	74.71	13.41	11.88			
2	44.2	1.1	69.78	12.48	17.74			
3	33.6	1.1	61.68	22.57	13.63	1.51		.61
4	48.0	1.1	54.83	28.11	17.06			
5	43.3	1.1	66.44	14.47	16.57		2.52	
6	23.5	1.6	62.43	23.39	14.18			

* IM is matter insoluble in HCl or HNO_3.

The analyses showed a larger percentage of aluminum than was intended, probably because the aluminum was put in last, melted very quickly, and seemed to fuse readily with the molten mass which was immediately poured into the molds without the aluminum having had much time to oxidize. The series formed by No. 1, 2, 6, and 4 had increasing percentages of manganese and aluminum, No. 3 had 1.5% lead and No. 5 had 2.5% of iron.

When the specimens had been prepared as given above, they were tested by the magnetometric method. The water jacket furnished an excellent means of controlling the temperature, as the hydrant water remained quite constant at 18° C. and this was the temperature of all the tests unless otherwise specified.

The data for the original test of a complete hysteresis cycle for No. 3 are given in Table 2 and shown graphically for the other specimens in Fig. 2 and 3. No curves are given for specimens 1 and 2, as they were so slightly magnetic as to be beyond the sensitiveness of the apparatus.

A study of the curves shows that No. 6 was the best of the series as it had the largest value of B with the least hysteresis, and Table 1 shows that it came nearest having the individual elements in the proportions of their atomic weights, i. e., copper 62.4%, manganese 23.4%, aluminum 14.2%.

TABLE 2

ORIGINAL TEST OF SPECIMEN NO. 3

I	*H*	*B*	*M*	*I*	*H*	*B*	*M*
+97	0.0	1220	....	—35	24.0	465	19.3
160	24.0	2030	84.6	91	34.4	1175	34.2
174	34.4	2220	64.6	167	51.6	2120	41.0
207	51.6	2640	51.2	216	68.8	2785	40.5
236	68.8	3030	44.1	188	51.6	2415	46.8
264	86.0	3400	39.5	153	34.4	1960	57.0
278	103.0	3595	34.9	132	24.0	1685	70.3
306	120.0	3960	33.0	62	0.0	778	.....
375	137.4	4860	35.4	+76	24.0	980	40.3
237	68.8	3040	44.2	132	34.4	1695	49.3
216	51.6	2760	51.5	195	51.6	2500	48.5
202	43.0	2580	60.0	229	68.8	2950	43.0
188	34.4	2395	61.6	......	....	....	......
174	24.0	2210	92.2	......	....	....	......
104	0.0	1305	....	......	....	....	......

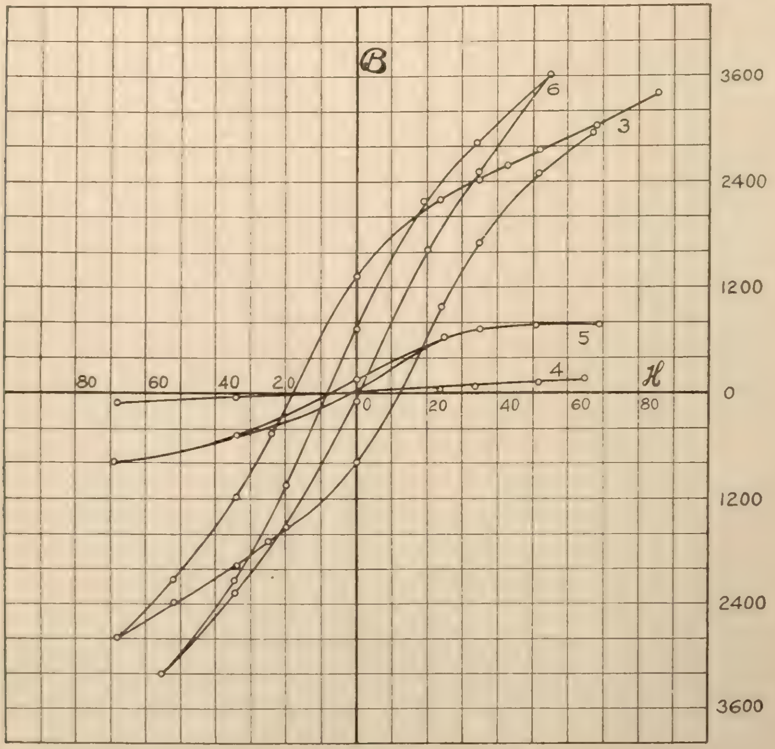

FIG. 2.

No. 3, with the lead in it, gave about three-fourths the value for B that No. 6 did and had much greater hysteresis. No. 5, with the iron in it, gave about one-third the value for B that No. 6 did, but had very little hysteresis. No. 4 which had a high per cent of manganese is only slightly magnetic.

For the purpose of comparison a few other tests gave results as shown below.

Good Swedish iron, H = 50, B = 12,000 to 14,000
Iron Wire, H = 50, B = 7,500
No. 6 alloy H = 50, B = 3,600

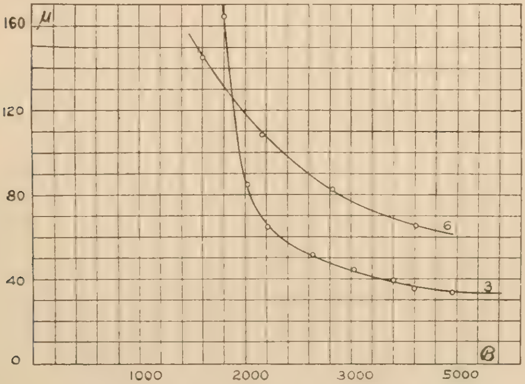

FIG. 3.

IV. TESTS

1. Working Hypotheses.—It was thought that in a way somewhat analogous to that of iron, the manganese, or some compound of it, in the Heusler alloys, might have two or more allotropic forms, which the alloying rendered magnetic at ordinary temperatures. Also, if the alloys were the most magnetic when the components were in the proportions of their atomic weights, it seemed possible that some compound might be formed which was the magnetic substance.

2. Preliminary Tests.—To test the first hypothesis, specimen No. 4 was cut into six pieces about 8 cm. long, placed in an Heraeus electric furnace and slowly heated. At each 100° interval, the temperature was kept constant for at least fifteen minutes, and then one piece was taken out and quickly quenched in water.

The theory of quenching is that the substance remains in the molecular condition that it was in at the temperature from which it was quenched.

It was found that when quenched between 300°C. and 400°C., the specimen was slightly magnetic; between 500° C. and 650° C., non-magnetic; and above 700° stronger than the original. These tests were simply qualitative, as the specimen was suspended by a thread and then placed between the poles of a horse-shoe magnet. The tests were repeated several times and were consistent. Quenching in the field of a Du Bois magnet of about 5,000 lines per square centimeter was also tried. It seemed to make no difference in the permanent magnetism, whether the pieces were quenched in the field or put in the field after being quenched.

Because the quenching in water cracked the specimens and made them very brittle, cooling in air was tried and found to produce the same result as quenching in water from a somewhat lower temperature.

3. *Effect of Quenching on Magnetic Properties.*—For more careful tests, the strongly magnetic pieces, No. 3 and 6, were used. They were heated to the temperature shown, in the order of the

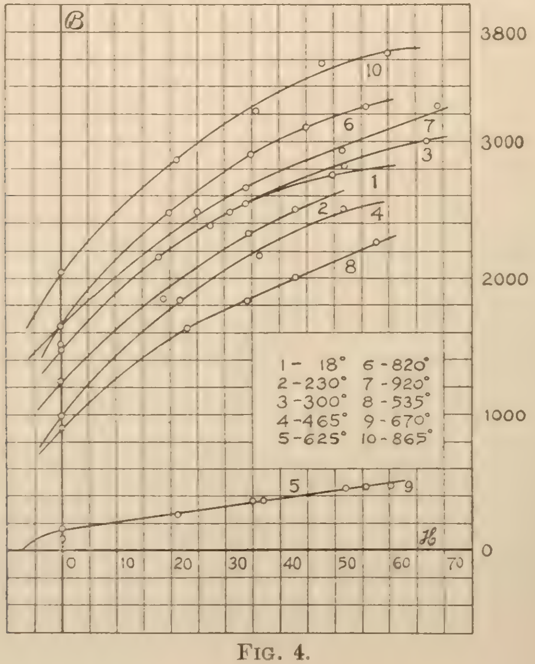

FIG. 4.

numbers of the curves, allowed to cool in air, and tested at 18°C. The results for No. 3 are shown graphically in Fig. 4, and the results for No. 6 are plotted in Fig. 5. In order to simplify the figures, the upper fourth only of the hysteresis curve is plotted from the data for the complete cycle.

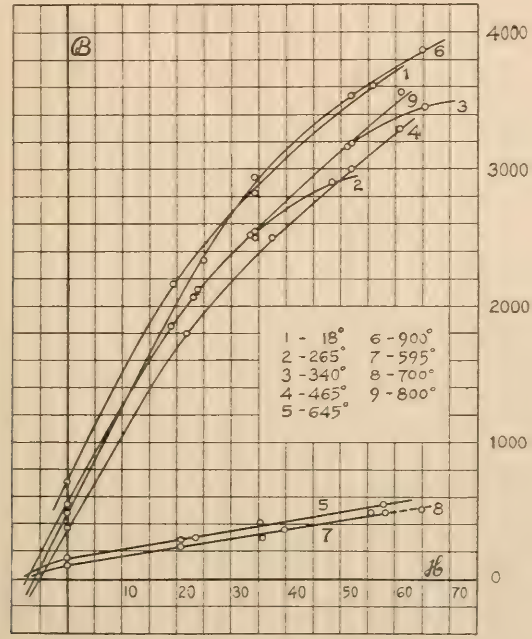

FIG. 5.

A study of the curves makes several generalizations possible. It is quite evident in both figures that when the specimen is heated to temperatures between 600°C. and 700°C. and allowed to cool in air, B is reduced to about 15% of its former value. On heating to still higher temperatures, B regains its former value and when quenched from within 100° of the melting point of the specimen, gives the highest value obtained. No. 3 has a lower value of B but a larger hysteresis and retentivity than No. 6. The retentivity increases in about the same proportion as B. To show the effect of quenching in another way, the values of permeability M, for H = 50, are plotted against quenching temperatures in Fig. 6.

Both curves show minimum values between 600°C. and 700°C. and larger values for the higher temperatures. The values of M around 250° C. are low. This is the temperature at which it has always been noticed that the beneficial effects of baking are lost.

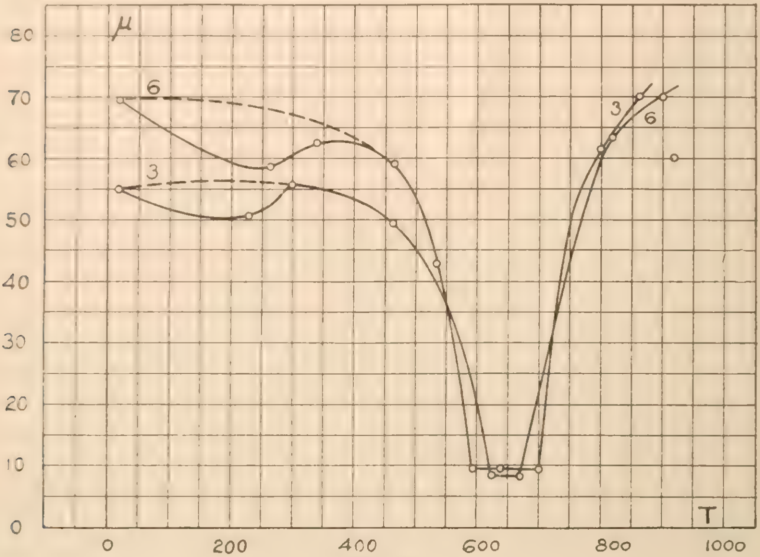

FIG. 6.

After the above series of tests of heating to a given temperaature and cooling in air, specimen No. 3 was heated to several high temperatures and quenched in water. As mentioned above, the effect was much the same as cooling in air, although in the latter process there would undoubtedly be time for a limited transformation, thus making the effective air quenching temperature somewhat lower than that actually read.

Quenching No. 3 in water after heating for 20 minutes at 870°C. gave the highest value ever observed for B.

For H = 69, B = 4,500, M = 65 and for H = 52, B = 4,300, M = 83.2.

These data seem to emphasize the point that the alloy is in its most magnetic state in the molecular condition existing near its melting point.

V. THERMAL ANALYSIS

It is evident that there is something peculiar about the condition of the alloy between the temperatures of 600°C. and 700°C., and in the hope of throwing some light on this and on the hypothesis that a chemical compound might be formed, a limited thermal analysis was undertaken. The general method of thermal analysis consists in heating the substance above its melting point, allowing it to cool under constant conditions, and taking time and temperature readings at regular intervals. From the form of the curve obtained, it may be possible to draw conclusions regarding the composition and molecular arrangement of the substance.

The particular method and some of the apparatus was that used by Clement and Egy[1] in the calibration of their thermocouples.

A cross-section of the electric furnace used by them is shown in Fig. 7. It has the heating coil on the inside of the furnace.

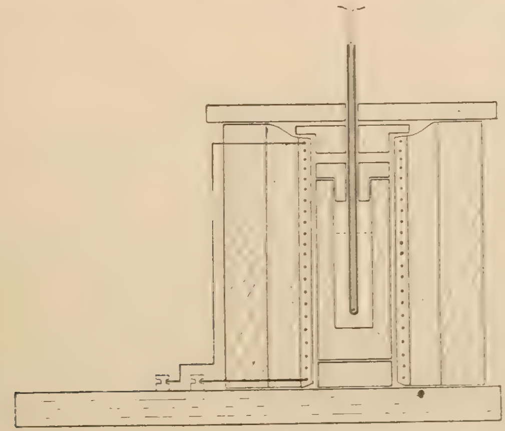

FIG. 7.

The coil of heavy nickel wire was wound on a sectional wooden core and plastered over with magnesite mixed with glue. When this had hardened, the core was removed and the coil put inside two cylinders of magnesite, the intervening spaces being filled with calcined magnesia. A large barrel of water with two iron grids for electrodes, the upper one being movable, was used as a control resistance for the furnace. The furnace would carry a maximum current of about 30 amperes on 110 volts.

The specimen for which a cooling curve was to be made was put in a graphite crucible and covered with powdered charcoal to give a reducing atmosphere. The crucible was made from a graphite rod turned to the required size in a lathe, and fitted rather closely inside the furnace.

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[1] University of Illinois Engineering Experiment Station Bulletin No. 36.

The thermocouple was enclosed in a quartz tube of 2 mm. internal diameter, the wires being insulated from each other by mica strips. This tube passed through holes in the lids of the furnace and crucible and was pushed down to within 2 cm. of the bottom of the crucible when the charge was melted.

The e. m. f. of the couple was balanced against that of a Weston standard cell by means of an Otto Wolff potentiometer arranged to be direct reading. This is shown diagrammatically in Fig. 8. The dials of the potentiometer were set on the numbers corresponding to the e. m. f. of the standard cell and the auxiliary resistance in series with the battery and potentiometer adjusted for a balance. This balance was sensitive to 1 ohm in 20 000 ohms.

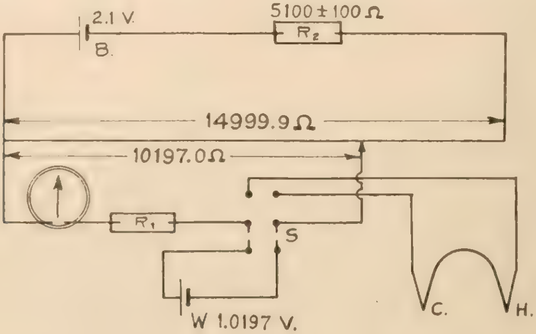

FIG. 8.

Thermocouple readings were taken in microvolts (MV) and the figure in the last place was found from the galvanometer reading. The resistance in series with the galvanometer and the scale distance were so adjusted that a deflection of one scale division corresponded to one microvolt. It was thus possible to read to 1 MV at 10 000 MV.

The thermocouple used was made of platinum-platinum, 10% rhodium, and was carefully calibrated at the freezing points of three pure metals which are accurately known—zinc 419°C., silver 961.5°C. and copper 1084°C.

When time-temperature readings are taken on a pure substance as it cools from the liquid state, the temperature of the substance falls with that of the furnace until the freezing point is reached. There it remains constant until all the substance is frozen and then quickly falls to that of the furnace again. It has been found experimentally that the freezing point is much more definitely marked than the melting point, and that it is much easier to keep conditions constant when cooling than when heating.

The calibration data for the couple used are given in Table 3.

The temperature was assumed to be a parabolic function of the e. m. f. as expressed by the equation

$$T = a + bE - cE^2$$

Table 3 gave three pairs of readings for T and E, from which the three constants in the equation were determined and a temperature-microvolt curve calculated. For this couple the equation was

$$T = 118.067 + 0.128478E - 2.40282 \cdot 10^{-6}E^2$$

For small intervals MV are approximately equal to tenths of a

TABLE 3

Copper Point		Silver Point		Zinc Point	
Time	MV	Time	MV	Time	M
5:16:00	9327	2:42:00	7982	2:21	2488
:20	9280	:20	7963		
:40	9218	:40	7923		
:17:00	9150	:43:00	7875	:22	2460
:20	9094	:20	7812		
:40	9058	:40	7752		
:18:00	48	:44:00	7688		
:20	52	:20	70	:23	2455
:40	51	:40	64		
:19:00	50	:45:00	63		
:20	50	:20	62	:24	55
:40	50	:40	59		
:20:00	49	:46:00	46		
:20	47	:20	7580	:25	55
:40	40	:40	7470		
:21:00	10	:47:00	7380		
:20	8940	:20	7300	:26	53
:40	8700	....	....		
:22:00	8594	....	....		
:20	8490	....	....	:27	2447
:40	8400	....	....	....	......

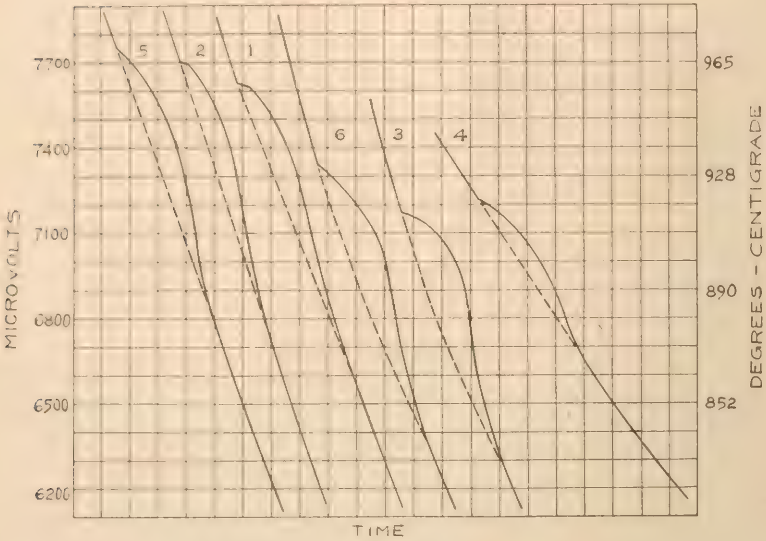

FIG. 9.

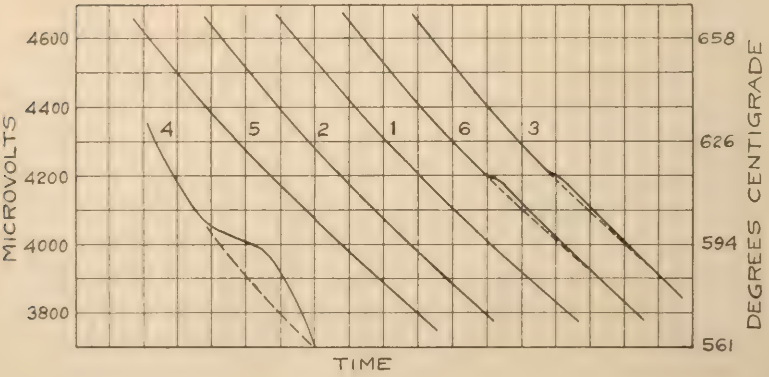

FIG. 10.

degree and it was thus possible to read to 0.1° at 1000°C., although the absolute value of the temperature was probably not closer than ± 1°.

The specimen of alloy to be tested was broken into small pieces, put into the crucible, covered with powdered charcoal, and heated in the furnace. It was kept above the melting point for

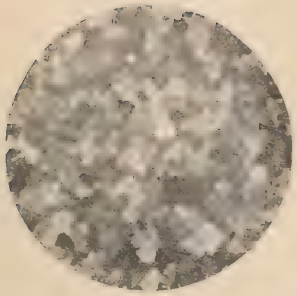

FIG. 11*. Specimen 1, twice recast, no further heat treatment. Etched with $FeCl_3$. Very slightly magnetic.

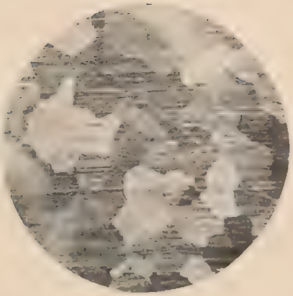

FIG. 12. Original specimen 1, heated for 1 hour at 880° and quenched. Etched with $FeCl_3$. Non-magnetic.

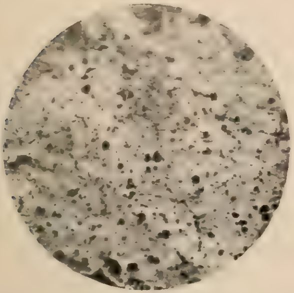

FIG. 13. Same as in FIG. 12, repolished and etched with HNO_3.—Blank pits are due to imperfect surface.

FIG. 14. Specimen 2, twice recast, no further heat treatment. Etched with $FeCl_3$. Very slightly magnetic.

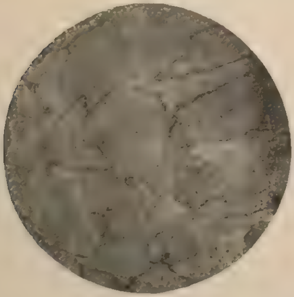

FIG. 15. Specimen 2 heated for two hours at 415° and quenched in water. Polished not etched. Very slightly magnetic.

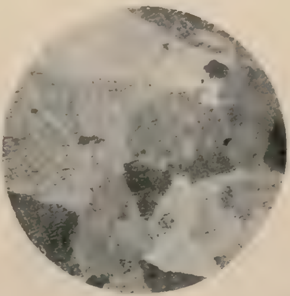

FIG. 16. Specimen 2 heated for 3 hours at 575° and quenched. Etched with $FeCl_3$. Very slightly magnetic.

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[*] All temperatures Centigrade.

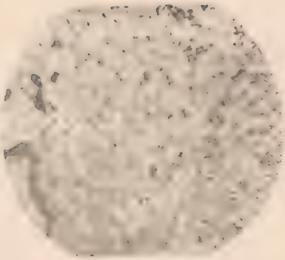

FIG. 17. Original specimen 3 heated for 1 hour at 345° and quenched. Polished but not etched. Strongly magnetic.

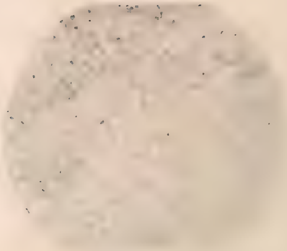

FIG. 18. Original specimen 3 heated for 4 hours at 425° and quenched. Polished not etched. Almost non-magnetic.

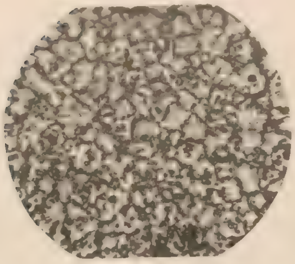

FIG. 19. Same as FIG. 18. Etched with HNO_3 and slightly repolished. Almost non-magnetic.

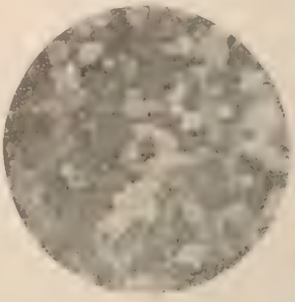

FIG. 20. Specimen 3, twice recast, no further heat treatment. Etched with $FeCl_3$. Strongly magnetic.

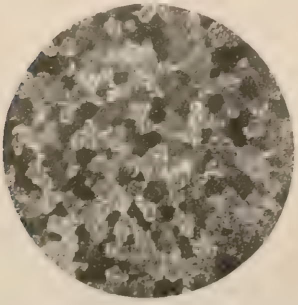

FIG. 21. Specimen 3 heated for 30 minutes at 466 and quenched. Etched with $FeCl_3$. Only slightly magnetic.

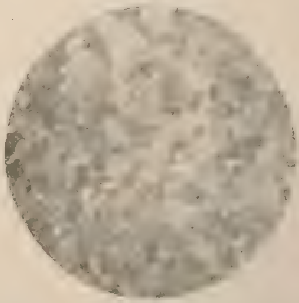

FIG. 22. Specimen 3 heated for 3 hours at 495° and quenched. Etched with $FeCl_3$. Strongly magnetic.

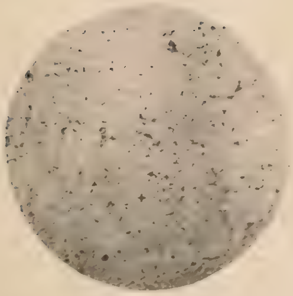

FIG. 23. Original specimen 5 heated for 2 hours at 620° and quenched. Etched with HNO_3. Slightly magnetic.

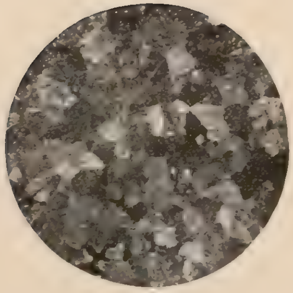

FIG. 24. Original specimen 5 heated for 2 hours at 880° and quenched. Etched with $FeCl_3$. Very slightly magnetic.

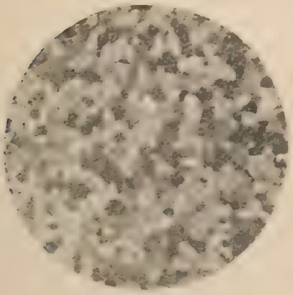

FIG. 25. Specimen 5, twice recast, no further heat treatment. Etched with $FeCl_3$. Slightly magnetic.

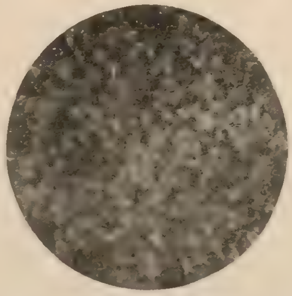

FIG. 26. Specimen 6, twice recast, no further heat treatment. Etched with $FeCl_3$ Strongly magnetic.

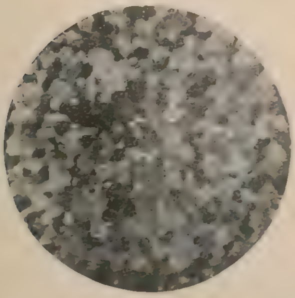

FIG. 27. Specimen 6 heated for 30 minutes at 470° and quenched. Etched with $FeCl_3$. Only slightly magnetic.

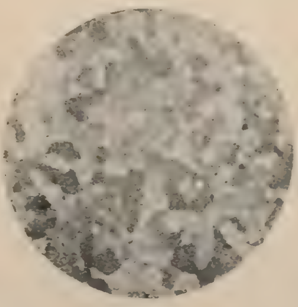

FIG. 28. Specimen 6 heated for 3 hours at 590° and quenched. Etched with $FeCl_3$. Strongly magnetic.

TABLE 4

COOLING CURVE DATA FOR SPECIMEN NO. 3
READINGS TAKEN EVERY 20 SECONDS

Time	MV	Time	MV	Time	MV
3:30	7628	3:45	5417	4:00	4192
	7572		5383		73
	08		46		53
:31	7440	:46	10	:01	38
	7376		5272		18
	10		40		4097
:32	7240	:47	04	:02	..
	7173		5170		.
	63		37		4040
:33	57	:48	03	:03	14
	50		5073		3996
	37		40		78
:34	.. ....	:49	07	:04	58
	08		4978		40
	7084		50		19
:35	52	:50	18	:05	00
	18		4888		3882
	6983		60		63
:36	36	:51	30	:06	44
	....		00		23
	6620		4773		03
:37	... .	:52	44	:07	3788
	6468		16		70
	00		4694		54
38	6338	:53	64	:08	35
	6280		37		18
	20		14		..
39	6172	:54	4587	:09	3684
	18		62		..
	6072		34		59
:40	24	:55	10	:10	42
	5980		4487		28
	42		63		12
:41	5893	:56	38	:11	3595
	55		14		80
	12		4392		67
:42	5773	:57	66	:12	52
	30		45		33
	5693		22		18
:43	50	:58	00	:13	00
	11		4280		3486
	5573		59		70
:44	30	:59	37	:14	54
	5490		19		37
	55		02		22

some time, to allow the pieces to fuse together, then the couple was pushed down into it, the heating current shut off, and readings of the potentiometer taken every 20 seconds. It took about two hours for the temperature to fall to 300°C.

When it was desired to pass very slowly through a point, the heating current was merely reduced, and, in order to keep it constant, the storage batteries were "floated on the line". The battery of forty cells was charged with a current of, say 16 amperes, while a current of 18 amperes was taken from it. This current would remain quite constant for several hours.

The cooling curve data for the six specimens are shown in Fig. 9 and Fig. 10, the curves in each figure being numbered to correspond with the specimen. Microvolts are plotted as ordinates and time as abscissas. The squares in the figure are centimeters and 1 mm. is equal to 10 sec. so the points are too close together to be shown in the figure.

VI. DISCUSSION OF COOLING CURVES

The upper parts of the cooling curves are plotted in Fig. 9 and the lower parts in Fig. 10. In the intervening region, the specimen cooled regularly with the furnace. The various curves show four characteristic regions, viz., (1) an almost straight line where the molten liquid is cooling with the furnace; (2) an abrupt change to an almost horizontal line, curves 1 and 2, Fig. 9, which indicates that a pure substance is freezing out of the solution; (3) a rounded portion with a quick drop to the temperature of the furnace which is given by the dotted line; (4) another abrupt change in slope in curves 3, 4 and 6, Fig. 10, which indicates a transformation, or recalescence point.

All the curves are those generally characteristic of solid solutions, but there are not enough data to draw any conclusions as to the formation of a compound. It was noticed that when an alloy was at the temperature represented by the bulge in its curve in Fig. 9, it was possible by pressing a rod into the crucible, to squeeze a liquid out of the semi-solid mass much as water can be pressed out of a sponge.

The data for each curve were taken at least twice and the freezing points found to be quite constant. The depressions of the freezing points are in the order of increasing percentages of manganese, except No. 5, which is high, possibly on account of the 2.5% of iron in it, and No. 3, which is low, due to the 1.5% of lead in it.

Curve 4, Fig. 10, has an exceptional transformation point. The test piece for it was made of the small pieces of No. 4 used in the preliminary quenching tests above. When the specimen had cooled to about 595°C., a pronounced change evidently took place, for there was a distinctly audible cracking, and on examination, the charge was found to be very loose in the crucible and the contraction had crushed the quartz tube containing the thermocouple.

Curves 1, 2 and 5 do not show any transformation points, but curves 3 and 6 do, and this point is at the same temperature, 615°C., from which these specimens were non-magnetic on quenching. It is evident then that at this temperature and for specimens of the approximate composition of No. 3 and 6, there is a molecular rearrangement accompanied by an evolution of heat, and that when kept in this state by quenching, the alloy is non-magnetic, but if it is allowed to cool slowly to room temperature, it goes over into a magnetic form.

VII. PHOTO-MICROGRAPHY

Since it is generally agreed that the magnetism of substances is due to their molecular arrangement and since even superficial examination of these alloys showed changes in crystalline structure with the heat treatment, a study by means of photo-micrography was carried out This method has not been very generally used and specific directions are hard to find, so it will be described here somewhat in detail. A general account of the art of photo micrography will be found in the Appendix.

The apparatus consisted of a Leitz "New Universal Photo-Micrographic Apparatus" fitted with an incandescent gas lamp, a microscope with a wide tube, and an opaque vertical illuminator. The magnification could be adjusted to any desired ratio by a suitable combination of eye-piece, objective and length of camera bellows. The magnification at which the accompanying photographs were taken was determined exactly by taking a photograph of a standard grating and counting the number of lines per cm. in the picture.

It is necessary that the specimen should have a plane surface and a high polish in order to give a flat field for the microscope. This was accomplished by fixing the pieces in a block of wood and grinding them on a series of rapidly rotating disks covered with emery paper of increasing fineness and then grinding by hand on a glass plate charged with emery. Emery sufficiently fine for this purpose was obtained by shaking "flour" emery in water and decanting the liquid with the particles in suspension after allowing it to settle for various lengths of time from one minute to twenty-four hours. This grinding gives a plane surface but covered with very fine scraches.

To remove the scratches the specimen must be polished. Great care is necessary not to destroy the plane surface or make more scratches by a little grit or dirt getting on the polisher. First, a felt wheel charged with "Tripoli" was used, then a cotton buffer, and finally the process was finished by hand by rubbing on a linen cloth stretched over a glass plate and charged with jeweler's rouge. This process might be much simplified and hastened by using a good mechanical grinder and polisher, but the final work by hand, though rather tedious, seems to give the finest surfaces. Excessive polishing should be avoided because the rapidly rotating polisher seems to cause the metal to flow over a thin surface layer entirely obscuring the structure.

To bring out the structure clearly, it is usually necessary to "etch" the specimen, i. e., to use a reagent that will attack some parts more than others. The particular etching fluid must be determined by the composition of the substance studied. For the Heusler alloys, a dilute solution of nitric acid or a strong solution of ferric chloride with possibly a little hydrochloric acid added, was used to bring out the various points.

In taking the pictures, artificial light diffused by a ground glass screen was used in preference to daylight because of its greater constancy and ease of control. Stopping down the light and using a longer exposure gave more detail and sharper contrast. The focusing was accurately done with a lens and a fine adjustment.

The plates used were Standard "Orthonon", which are orthochromatic and non-halation. Because of the double film, the plates were developed slowly and until almost black, and left in the hypo and washed about twice as long as ordinary plates.

A printing paper with a glossy enameled surface was used to bring out the contrast in the structure. This glossy paper usually requires a special non-abrasion developer.

PHOTO-MICROGRAPHS OF THE HEUSLER ALLOYS

A large number of photographs were taken under various conditions and a few, which are all of a constant magnification of 50 diameters, are given in Fig. 11 to Fig. 28. The "original" specimens are pieces of those first cast and whose composition is given in Table 1. The others were made from the original ones of the same number by melting and recasting twice. This produced more homogeneous alloys with better mechanical properties. There was some slag formed at each melting, so the composition is probably not quite the same as the original, but they did not differ much, as the slag was of approximately the same composition as the specimen and was magnetic.

After a specimen had been heated and quenched, it was ground off for at least 2 mm. below the surface and care was taken to keep it from heating too much during the grinding.

The magnetic testing for this part of the work was done qualitatively by suspending the specimen by a thread in the field of a large permanent magnet. Slightly magnetic means that the specimen would orient itself with the field and possibly turn through an angle of 90° as the magnet was rotated. Strongly magnetic means that the specimen could be lifted by the magnet.

Some of the photographs show the three types of crystals described by Knowlton[1], but no such simple relation as he found between the size of the white crystals and the magnetization could be determined. In fact there seemed to be no essential difference between specimens 3 and 6 when quenched from the temperature that made them non-magnetic and when taken in their originally strongly magnetic condition. Also it would be difficult to distinguish between Fig. 11 and Fig. 22, although the former is of specimen 1 which was almost non-magnetic and the latter is of specimen 3 which was strongly magnetic.

The photographs of specimen 2 show a peculiar arrangement of long slender crystals pointing towards the center of the specimen where the crystals are smaller and more regular. The recast specimens all show much smaller crystals than the original ones, but none of them shows the pronounced effects due to quenching or annealing that were expected.

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[1] Phys. Rev., v.30, p. 123, 1910.

VIII. SUMMARY

The results of this investigation may be summarized as follows:

(1) The best specimen had the manganese and aluminum in the proportion of their respective atomic weights. It gave a maximum value of $B = 4000$ and $M = 57$ for $H = 70$, but had not reached saturation.

(2) Heating to various temperatures, cooling in air and testing at room temperature gave, for $H = 50$, a practically constant value for the permeability from room temperature to 300°, a rapid drop between 300° and 600° to about one sixth its former value, and beyond 700° a rapid rise to better than the original value at 900°. This series of changes was reversible for at least three times.

(3) The largest values of B and M were obtained when the specimen was quenched in water from near its melting point.

(4) A series of cooling curves gave melting points between 910° and 970°C., and curves characteristic of solid solutions.

The curves for the two magnetic specimens showed transformation points, with evolution of heat, at 615°C. Specimens quenched from this temperature are non-magnetic when tested at room temperature.

(5) It seems evident that there is an allotropic form of the alloy or at least of the magnetic constituent of the alloy, which has a non-magnetic molecular arrangement at about 600°C.

(6) A series of photo-micrographs showed differences in the crystalline structure due to previous history, but no simple relation could be established between the magnetization and the presence of certain crystals.

APPENDIX

PHOTO-MICROGRAPHY

The possibility of taking a photograph through a microscope was demonstrated as early as 1840, but only within the last two decades has it been possible to obtain satisfactory apparatus at moderate cost or to find books from which one could acquire a knowledge of the technique in a reasonable time. Even yet in this country photo-micrography is not so well understood or so fully applied as its advantages would warrant if better known.

It is the purpose here to give a few fundamental principles and essential points in the selection and use of the apparatus with some general methods of attack in the solution of the problems involved. Incidentally, it is well to distinguish a photo-micrograph, which is a photograph of a microscopic object, from a micro-photograph, which is a microscopic photograph of a large object.

Photo-micrography may be applied in almost any case where the microscope is used and in some cases, as will be shown later, this art has extended the field of the microscope. As a record of the objects seen in a microscope, a photograpb has advantages over the usual drawing, in that it tends to eliminate the personal bias and to give increased weight to the evidence presented. It saves much time when the necessary equipment and skill are once acquired, and it may be reproduced indefinitely in the positive print or the lantern slide. It is possible by the use of ultraviolet light to photograph objects that are invisible with ordinary light; by the proper choice of plates and color filters, the brilliant shading of colored objects may be correctly reproduced in monotone; because of the cumulative effect of the light on the sensitive plate, an object but dimly seen may be plainly photographed by prolonged exposure; while by the use of the cinematograph, moving pictures of a microscopic world may be had. Some application of photo-micrography is certain to be an advantage in almost any line of modern scientific work, and to the amateur photographer, it opens up new possibilities for the application of his highest skill.

APPARATUS

The apparatus necessary for photo-micrography depends upon the grade of work to be done, but it must include a microscope, a camera, a rigid support for them, properly prepared specimens, some source of illumination with means of control, suitable plates, printing paper, developing and fixing solutions. Various home-made combinations may serve, in some instances, but for the best work, particularly at high powers, special apparatus will be more satisfactory because of the corrections of the optical system and the devices for accuracy and ease of adjustment that are so great a saving of time and patience.

Microscope.—A special microscope is not necessary, but the one used should have a wide tube for photographing without the ocular, a stable base, an adjustable stage, and the best possible lens equipment.

There are two properties of lenses that must be especially considered in photo-micrography. The spherical aberration is due to the fact that monochromatic light is unequally refracted in passing through a single spherical lens, being refracted more near the edge than it is near the axis, so that all the rays incident on the lens cannot meet exactly in the same focus. Chromatic aberration is due to the fact that the lens acts as a prism, resolving the white light into spectral colors which are unequally refracted, and thus the different colors cannot be brought to a focus at the same point. By a suitable combination of a bi-convex crown glass lens with a plano-concave flint glass (one with double the dispersive power) the chromatic aberration can be corrected for any *two* colors, but there is still a secondary spectrum left because the dispersion of the colors is not proportional in the two kinds of glass. Also the spherical aberration can be corrected for any *one* color, usually that of maximum visual intensity, but there remains under-correction for the red and over-correction for the violet. A lens of this type is called an achromatic lens. As is well known, the light that is most active photographically on the ordinary sensitive plate, the so-called "actinic" light, is that which is near and beyond the violet end of the spectrum, while the eye is more sensitive to the longer wave lengths in the yellow-green. Thus the ordinary achromatic lenses with corrections for spherical aberration in the region of maximum visual intensity and for chromatic aberration between the red and violet are not the best for photo-micrography except under special conditions.

After an exhaustive investigation by Dr. Abbe, with the co-operation of the optical workshops of Zeiss and the German government, crown and flint glass were produced of such quality that, when combined with a fluorite lens and special eye-pieces, it was possible to correct the chromatic aberration of the objective for *three* colors, thus removing the secondary spectrum, and to correct the spherical aberration for *two* colors and therefore practically for all. These lenses are so well corrected over the whole visual and photographic range as to justify the name apochromatic. Besides their almost perfect achromatization, these apochromatic lenses have the advantage of utilizing the full theoretical aperture, and of permitting a great increase in magnification by the use of special high power oculars. These lenses are especially valuable for photo-micrography.

Camera.—The camera should have a stable base and a means by which the microscope may be firmly clamped to it. A very frequent cause of indistinctness in the negative is the vibration of the supporting table. This may at least be diminished by having the camera and microscope vibrate synchronously, and it may even then be necessary to work at some time when the whole building is quiet. Another important point to be observed is that the guides and grooves for the plate holder and ground glass slide should be accurately adjusted so that the plane of the photographic plate will lie exactly in that of the glass on which the object is focused. For the final adjustment, a clear glass and a focusing lens should be used instead of the ordinary ground glass. The extension of the camera determines the enlargement, but it is the microscope that determines the *definition:* the photograph may be merely of a convenient working size. The power of the objective used should be as low as is consistent with satisfactory definition, because this is the best working condition for the microscope, and the difficulties in the photographing increase rapidly with the magnification. To obtain the magnification of a given arrangement, the simplest method is the direct measurement of the enlarged image of a micrometer microscope ruled in thousandths of a centimeter, and an analogous method is applicable in determining the actual size of an object.

Illumination.—The use of artificial light for photo-micrography is almost universal because of its constancy and ease of control; and the choice between kerosene, gas, calcium, or electric light depends upon their availability and upon the particular work.

The yellow kerosene flame, though weak, may prove very satisfactory at low magnifications with ordinary achromatic lenses and ortho-chromatic plates without a color screen, while the arc light is far more intense, rich in actinic rays, and, in general, requires apochromatic lenses and color screens. Before making an exposure, it is necessary that the light be properly focused and centered, and the field properly illuminated. This involves centering the condenser, and then centering the light and focusing the condenser until a sharp image of the light source is projected through the condenser into the plane of the object. To secure uniform illumination, it is often necessary to introduce a bull's eye condenser between the light source and the sub-stage condenser at such a point that the light will be at the principal focus of the bull's eye, thus giving parallel light. When a ground glass is used to give diffused light or a colored glass for colored light, it may be placed between the bull's eye and the condenser. The light from the microscope is then centered on the ground glass of the camera, and the circle of light should be uniformly illuminated and sharp around the edges. For instance, if there is a dim halo around the image, it may be due to a flooding of the objective with too much light, which should be reduced by means of the diaphragm in the condenser; or, on the other hand, it may be due to the use of an objective with too narrow an aperture.

Preparations.—It is assumed that the best specimens will be photographed, and the inherent difficulties of the process should not be increased by faulty technique in the preparation. In general, a flat field with clearly contrasted structure, is desirable. The depth of vision with which an object is seen in a microscope is the sum of two factors; the focal depth of the objective and the accommodation of the eye. This latter factor is absent in the photographic plate, so sharp definition is obtained only in the critical plane. No such composite image is possible as the microscopist often obtains visually by small alterations in the focus; hence the necessity for a flat field and a sharp focus.

Sensitive Plates.—As noted above, the ordinary plate is not sensitive to red and yellow light, but plates may now be obtained that are more or less sensitive to these colors and are variously called orthochromatic, isochromatic and panchromatic, depending upon their particular qualities. It is necessary that achromatic lenses be especially corrected for photography and that orthochromatic plates and often a color screen be used with them. With apochromatic lenses, ordinary plates may be used although the plates that are sensitive over the wider range are better. The red-sensitive plates must be handled in deep ruby light or total darkness. Another important correction is that for halation —a blurring of the image due to light diffusely reflected from the back of the glass plate. The manufacturers correct this by giving the plate an extra coating of absorbent film. Any plate may be rendered non-halation by "backing". A coating of caramel, colored with lamp black, is recommended. It may be applied to the glass side of the plate with a stiff brush and removed with a wad of damp cotton before development. In general, the plate should be developed for contrast and detail with considerable density. Hydroquinone and metol are satisfactory developers.

Exposure.—The time of exposure depends on so many variables that exact directions can not be given, but the following rule will be found valuable: Secure as satisfactory an arrangement as possible before exposure, determine the time of exposure by trial, and then keep the conditions constant, or at least vary only one at a time so that the effect can be calculated. This rule applies to light source and distance, magnification, plates, developer, printing paper and time of development and printing. To determine the exposure by trial, withdraw the dark slide one cm. at a time at 10-sec. intervals thus obtaining a series of strips of known exposures. When developed for an average time, there is no difficulty in determining which is the best. The development of the plate gives much information in regard to the correctness of the exposure. Although, in special cases, a negative may be intensified or reduced, it is generally better to repeat the exposure until a satisfactory negative is obtained.

The Positive.—The positive print may be either on paper or on a lantern slide. If $3\frac{1}{4} \times 4\frac{1}{4}$ plates are used, they will be very convenient for printing slides by contact. Because the light shines through the lantern slide, thus giving the variations in shading with all the wealth of detail, it is really the best kind of a positive. For printing paper, a glossy surface especially sensitized for contrasted effects, is essential and the prints may then be burnished or squeegeed.

Color Photography.—The modern idea of color photography is not to reproduce in the negative or postive print the actual colors of the object, but to get in monotone the shading which we see in the colors. If one wishes to photograph a relatively weak color with increased contrast, the general rule is to use a plate but slightly sensitive to that color, and a screen cutting off the color of the preparation. Under these conditions, the object will photograph as though it were black. On the other hand, if it is desired to decrease the effect of a certain color or "hold it back", a light of the same color should be used and a plate sensitive to that color. Thus if a red flower were photographed with a red light and a red sensitive plate, the negative would be strongly affected, but the positive would be light.

The color screens may be made of a glass cell filled with various colored liquids of different densities, or they may be made of colored glass. Several of the dry plate manufacturers have a series of screens spectroscopically adjusted to work with certain plates.

Ultra-violet Light.—The use of ultra-violet light in photomicrography has been one of the most beautiful developments in this science. A short theoretical discussion will be necessary for the proper appreciation of this point. The resolving power of a lens, i. e., its property of showing two points close together as distinct and not fused into one, is, for central lighting, directly proportional to the numerical aperture and inversely proportional to the wave length of light used. The numerical aperture in turn is directly proportional to the index of refraction of the glass and the sine of half of the angle included between the lines drawn from the principal focus to the ends of a diameter of the lens. The numerical aperture for dry mountings in air must always be less than unity, and is limited in practice to 1.50 for the best oil immersion. Thus there is a practical mechanical limit to the resolving power of an objective with any of the kinds of glass now available for lenses. Now the resolving power might be increased by decreasing the wave length of the light used, but the eye is not sensitive much beyond the violet, and, too, ordinary lens glass is opaque to the short wave lengths. Fortunately, fused quartz is transparent to ultra-violet light and the photographic plate is sensitive to it. By making all the lenses of quartz glass and producing the light by sparking between cadmium electrodes, the resolving power of a microscope may be doubled. The light from the cadmium spark has a wave length of .275 μ as compared with 0.589 μ for yellow sodium light, and is unusually monochromatic, thus eliminating the necessity of correcting for chromatic aberration. It is found also that many organic objects that are colorless under white light act as if they were colored under the ultraviolet. To render the object visible for selection and focusing, a uranium glass is used that fluoresces under the action of the ultra-violet light. The equipment for this work is rather expensive, on account of the extra apparatus necessary to produce the light, and the cost of the quartz lenses, which are monochromatic and not available for other work.

APPLICATIONS OF PHOTO-MICROGRAPHY.

Since the biologist has always made the largest use of the microscope, much of the technique of photo-micrography has been developed for his particular use and for the text-book written from his point of view, but there are other important fields of application of which two will be discussed by way of illustration.

Metallurgy.—Chemical analysis will give the percentage composition of a substance, but sometimes a wide difference in physical properties is found in substances having the same chemical composition. The application of the modern theory of solutions to solids has led to a remarkable development in our knowledge of alloys, and in this work thermal analysis and microscopic examination have been the chief aids to chemical analysis. It has been found that such physical properties as hardness, tensile strength or thermal conductivity are functions of the crystalline structure which may be seen in the microscope when the specimen is properly prepared. In the case of our best known alloy, steel, the physical properties of any given specimen are largely determined by the heat treatment, and the microscope shows that this treatment is accompanied by a definite change in structure. In the investigation described in this bulletin, an attempt was made to find a relation between the magnetic properties and the structure in the Heusler alloys, and although this was not established, there evidently is a relation between the structure and the heat treatment. When the structure associated with particular properties has once been established, a microscopical examination is a rapid and convenient test for these properties, and the photograph is a good method of recording the data.

The technique for the study of metals differs in several points from that for transparent substances. Because the metals are opaque, it is necessary to use reflected light. With long focus objectives, the illumination may be from the side, but should be as nearly vertical as possible to prevent loss by reflection. For higher magnifications, some form of vertical illumination is used. It consists generally of a prism or glass plate inserted in the microscope tube above the objective, and set at such an angle that it reflects the light received through an aperture in the side of the tube down through the objective on to the specimen. A point to be noted is that the insertion of this illuminator increases the tube length and must be compensated for in the sliding sleeve as the lens system is corrected for a given length of tube, usually 170 mm. The common microscope objectives are corrected for the thickness of the cover-glass, and hence for work with dry mountings, it is necessary to have objectives without this correction, or with a device in the objective to compensate for it.

The application of photo-micrography to the study of the Heusler alloys has been given in detail on page 23 of this bulletin, and illustrates the general methods, though each alloy requires some special treatment. With alloys of better mechanical properties, the grinding and polishing are much simpler processes than those described. There seems to be no definite rule by which one can determine the best etching agent, and a large number of acids, bases or salts may have to be tried before a satisfactory one is found. Sometimes beautiful oxidation effects can be produced by heating, and indeed, it is difficult to prevent these effects at ordinary temperatures. If the specimens are to be preserved, they may be coated with a very thin transparent varnish.

Geology.—The microscope is in common use in geology, and a series of photo-micrographs gives an unsurpassed method of comparing a number of specimens. The size and shape of rock crystals not only furnish a means of recognizing the kinds of crystals, but may enable one to estimate the temperature at which they were formed and the rate of cooling, problems of particular importance to the dynamical geologist. Many specimen can be ground thin and viewed by transmitted light and others may be seen by side or vertical illumination. The use of polarized light brings out the anisotropic optical properties and shows the axial placing of the crystals. The color effects with polarized light are most beautiful. By a suitable combination of transmitted light, reflected light and dark ground illumination with long focus planar objectives, one may obtain photographs of minute fossils that show both the external form and a certain amount of internal structure.

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RESISTANCE TO FLOW THROUGH LOCOMOTIVE WATER COLUMNS

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RESISTANCE TO FLOW THROUGH LOCOMOTIVE WATER COLUMNS.

1. *Preliminary.*—Water is delivered to the tanks of locomotives either directly from an elevated tank or through a pipe line and a discharging device known as a water column, standpipe, or penstock. Qualifications commonly recognized as desirable in a locomotive water column are (1) the hydraulic properties of rapid discharge, moderate frictional resistance to flow, easy movement and control of valve, and freedom from objectionable water hammer; (2) a mechanical construction permitting easy and satisfactory operation and convenient inspection and repairs; (3) a design giving reasonable cost of manufacture and maintenance. With the increased importance of time in train service the importance of the first-named element in a water column is becoming recognized. Little definite information on the hydraulics of water columns has been available, so that in designing water service plants the engineer has been largely in the dark in making allowance for the head lost through the water column or even in the selection of size of water column, and the differences of practice are considerable. The American Railway Engineering and Maintenance of Way Association has appreciated the importance of having fuller information on the flow through water columns. Growing out of this interest, this investigation was taken up in cooperation with the Committee on Water Service of the Association.

2. *Scope of Bulletin.*—This bulletin gives the results of tests, made in the Hydraulic Laboratory of the University of Illinois, of fourteen water columns covering the principal types used in supplying water to locomotives in the United States. The tests were made for the purpose of determining the losses of head through the column, including the loss in the valve and in the riser and spout. Incidentally, the action of the valve in closing and the changes in pressure in front of the valve were studied with a view of getting information on the influence of the type of the valve on the development of water hammer in the supply main. A general statement of the principles of water hammer and their application to a water service installation, as well as of

relief valves, is also given.

There are also given in the Appendix diagrams for calculating the loss of head in pipe and diagrams for the loss of head in elbows, tees, and valves. These are presented with the thought that their inclusion here will be a convenience to a great many readers.

3. *Acknowledgment.*—Acknowledgment is made to the Committee on Water Service of the American Railway Engineering and Maintenance of Way Association, and especially to Mr. Robert Ferriday, Engineer Maintenance of Way, Cincinnati, Cleveland, Chicago, and St. Louis Railway, for cooperation and for arranging with the manufacturers for the use of the water columns tested, and for valuable suggestions and helpful interest throughout the tests. Acknowledgment should also be made of the fact that the Water Service men of the Peoria and Eastern division of the C. C. C. & St. L. Ry. assisted in setting up the water columns. The results of a part of the tests were embodied in a report to the American Railway Engineering and Maintenance of Way Association.

4. *Water Columns.*—The water columns were furnished by the manufacturers as representing the regular article supplied to the trade. The 10-in. size was chosen for the tests, since it is a medium size and since with this size a high velocity could be maintained through the water columns with the facilities of the Laboratory. The columns were set up by experienced water-service men. As a representative of the manufacturing company furnishing the water column was present for the types with which the men were not familiar, the proper erection of the water columns was insured.

The water columns will be designated by the Roman numerals given in Table 1, the numbers being used in the order in which the tests were made. The Poage water column is manufactured by the American Valve and Meter Co.; the Otto, by the Otto Gas Engine Co.; the Gulland, by the Best Manufacturing Co.; the Mansfield and the U. S., by the U. S. Wind Engine and Pump Co.; the Sheffield, by Fairbanks, Morse & Co.; and the Golden-Anderson, by the Golden-Anderson Co. The Sellew is a new water column designed by Mr. W. H. Sellew, Principal Assistant Engineer of the Michigan Central Railroad, and now handled by the Cleveland Railway Supply Co.; in the tests with the Sellew the experimental valve replaced the valve of the Gulland water column. In the classification, the terms globe, angle, and gate are used as roughly indicating the form of the valve. The method of operating the valve, both in the manual type and in the hydraulic type, differed considerably. The form of spout used was chosen by the manufacturer, the form used being no part of the test, since both rigid and adjustable spouts are made by manufacturers. The nozzles marked "anti-splash" are cylindrical nozzles having a reticulated diaphragm to straighten the filaments of the discharging stream. The one marked "conical anti-splash" was an open cone. The other nozzles were plain cylinders.

The tests of the first ten water columns were made in October, November, and December 1909 and the results were reported to the Committee on Water Service of the American Railway Engineering and Maintenance of Way Association. Water column XI was tested in May 1910. Water columns XII, XIII and XIV are new forms of columns, designed after the first tests were made; these were tested in January 1911.

The make-up of the water columns is shown in the cuts, Fig. 14, 16, 18, 20, 22, 24, 26, 28, 30, 32, 34, 36, 38, 40 and 42 at the end of the text.

TABLE 1.

CLASSIFICATION OF WATER COLUMNS.

Designation	Name	Valve	Operation	Spout	Nozzle
I	Poage, style D	Globe	Manual	Telescopic	
II	Poage, style B	Globe	Manual	Rigid	Anti-splash
III	Poage, style F	Angle	Manual	Telescopic	
IV	Poage, style A	Angle	Manual	Rigid	Anti-splash
V	Otto	Angle	Manual	Ball & Socket	
VI	Gulland	Globe	Hydraulic	Rigid	Anti-splash
VII	Mansfield	Angle*	Manual	Rigid	With appended bucket
VIII	U. S.	Angle	Hydraulic	Rigid	With appended bucket
IX	Sheffield	Angle	Combination	Rigid	Conical anti-splash
X	Golden-Anderson	Globe	Hydraulic	Rigid	Anti-splash
XI	Sellew	Gate	Hydraulic	Rigid	Anti-splash
XII	Poage, style H	Angle	Manual	Telescopic	
XIII	Poage, style H	Angle	Manual	Rigid	Anti-splash
XIV	Otto	Angle	Manual	Telescopic	

*This angle valve does not replace the elbow.

5. *General Arrangement of Apparatus.*—In a test, the water column was connected to a tank or standpipe (hereinafter referred to as the tank) as shown in Fig. 1; a constant head was maintained in the tank during a test run by means of large pumps; the rate of discharge of the water column was measured by means

of a weir; and the pressures were determined by mercury gauges. The tank was 4 ft. in diameter and 60 ft. above the foundation and acted as a regulating chamber in maintaining a uniform flow. The pipe B leading from the 15 in. outlet of the tank was 12 in. in diameter. The nipple at C was connected with a 10 in. pipe 3 ft. long, and the water columns were connected to this 10-in. pipe at E. The piezometer connection at D was from 6 in. to 15 in. from the inlet E of the water column according to the form of connection with the water column. One opening was placed in the side of the 10-in. pipe and one in the bottom, cocks being arranged to permit readings to be taken from the separate openings. A $\frac{3}{4}$-in. pipe led from the piezometer connection to a mercury gauge placed in a convenient position. A second piezometer connection was made in the vertical part of the water column at a point G (Fig. 1) about 6 in. below the usual ground line and this was connected with a second mercury gauge. The point G was chosen where conditions would be fairly common in all the

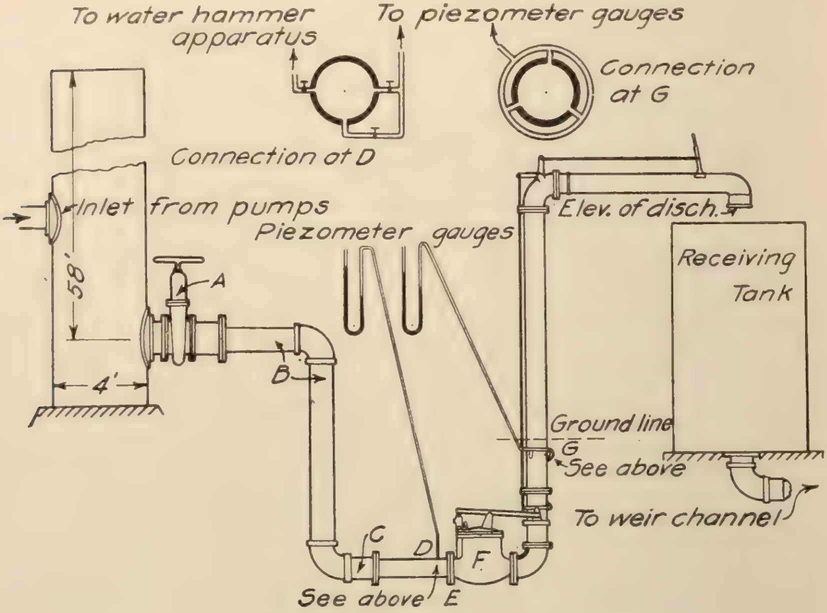

FIG. 1. GENERAL ARRANGEMENT OF APPARATUS.

water columns, being well above the valve arrangements in every case. The arrangement of the piezometer connection at D and at G is shown in the upper part of Fig. 1. The piping to the gauges was so arranged that any accumulating air would be carried out and the correct pressures observed. The effect of the elbows and the 10-in. connection was such that the distribution of flow over the section at D (Fig. 1) was not uniform, and this lack of uniformity may affect the piezometer readings noticeably for the higher flows, although observations taken alternately with the two piezometer openings at D showed only little differences. The difference in level between the zero level of these gauges and the discharge level of the water column was measured, and the pressure with reference to this discharge level was recorded. The level of the free discharge was considered as at the end of the spout, except that for the water columns with telescopic spouts (I, III, XII, and XIV), the center of the orifice discharging into the spout was used since there was free admission of air at this point. Having the observations and measurements, the total head lost in the column could be found. The observations also permitted the finding of the amount of head lost between the inlet to the column and the point in the riser just below the ground line; this would include the loss in the tee, the loss through the valve, and the frictional and other losses in the lower portion of the riser. The loss between the point below the ground line and the discharge outlet was also determinable. In all cases, of course, the head required to lift the water to the level of the spout was not counted as lost head. The discharge of the water column was measured over a 3-ft. standard weir having suppressed end contractions. This weir has been calibrated so that its coefficients are known with reasonable accuracy, and the conditions of the test were such that the measurement of the discharge is trustworthy.

6. *Procedure of Tests.*—For each of the water columns, tests were made in such a way as to give discharges of approximately, say, 1500, 2000, 2500, 3000, 3500, and 4000 gallons per minute, the head of water in the tank being made such as to produce the desired discharge. In starting the test the valve was opened under a low or medium head, and the water was maintained at a uniform height in the tank during the run. After the flow over the weir had become constant at about the desired discharge, readings of the pressure gauges and the hook gauge were taken

at four to five minute intervals for a period of twenty to thirty minutes. The pumps were then speeded up and the head in the tank increased until a quantity approximating the next higher discharge desired was obtained. After the flow over the weir had been found to be constant, readings were taken in a similar manner for a period of twenty to thirty minutes for the second run. Another increase in the head in the tank was then given and a run at the next higher discharge was made. The discharge was increased until it reached the limit of the capacity of the pumps for the head taken by the column under test. Repetitions were obtained to give a check on the general conditions of any test.

7. *Results of the Tests.*—The results of the tests are platted to logarithmic scale in Fig. 15, 17, ...41 at the end of the text. The abscissas give the discharge in gallons per minute and the ordinates the head lost in feet of water. The values of the total head lost through the water column in the several experiments are shown by open circles. The values of the head lost through the part of the water column above the ground line piezometer connection are shown by solid circles. In each case the head necessary to give the velocity head and the entrance head are, of course, not included.

8. *Loss Diagrams.*—The lines for the 10-in. water columns drawn on Fig. 15, etc., agree very closely with the observed results. It would seem that this line may be used with fair accuracy up to a discharge of 5000 gal. per min. As drawn, the head lost is made to vary as the square of the velocity. It appears that the variation as the square of the velocity fits the results of all of the tests as closely as any power which could be selected.

From general considerations, it would seem reasonable to expect that 8-in. and 12-in. water columns will have losses approximately the same as those from the 10-in. water columns at the same velocity of flow, although, of course, the element of pipe friction through the column would probably differ somewhat from this, and result in a somewhat smaller loss for the 12-in. columns and a larger loss for the 8-in. water columns. The lines marked "8-in. water column, estimated", and "12-in. water column, estimated", are based on the assumption of equal losses at the same velocities. Some error is doubtless involved in this assumption, but these quantities may be found useful until more accurate information is available.

Curves representing the total loss in head for the various types of water columns are platted to natural scale in Fig. 2, page 10. Curves representing the loss in riser and spout are platted in Fig. 3, page 11.

9. *Tables.*—A comparison of the loss of head in the several water columns at equal discharges may be of interest.

Table 2 gives the loss of head in the 10-in. water columns for discharges of 3000, 4000, and 5000 gallons per minute. The column marked "Valve" in the table gives the loss from the inlet to the ground line, and, of course, includes other losses than valve losses. The column marked "Riser" gives the losses from the ground line up, including that in the spout in water columns having fixed spouts. For Table 2 and the succeeding tables, the values were taken from the lines of the diagrams.

TABLE 2.

Loss of Head in 10-in. Water Columns.

The loss is given in feet of water for the discharge indicated.

Designation	3000 gal. per min.			4000 gal. per min.			5000 gal. per min.		
	Valve	Riser	Total	Valve	Riser	Total	Valve	Riser	Total
I	12.6	2.9	15.5	22.5	5.2	27.7	35.0	8.0	43.0
II	12.6	5.1	17.7	22.5	9.0	31.5	35.0	14.0	49.0
III	11.3	2.9	14.2	20.3	5.3	25.3	31.3	8.0	39.3
IV	11.1	5.1	16.2	20.0	9.0	29.0	31.0	14.0	45.0
V	6.4	6.1	12.5	11.4	10.9	22.3	18.0	17.0	35.0
VI	16.3	7.2	23.5	29.0	12.8	41.8	45.0	20.0	65.0
VII	20.0	6.1	26 1	35.7	10.8	46.5	55.0	17.0	72.0
VIII	9.0	4.0	13.0	16.0	7.0	23.0	25.0	10.0	35.5
IX	6.4	6.7	13.1	11.3	12.0	23.3	17.8	18.7	36.5
X	13.5	6.8	20.3	24.4	12.0	36.4	37.5	19.0	56.5
XI	1.6	7.2	8.7	2.8	12.6	15.4	4.3	19.7	24.0
XII	6.1	2.8	8.9	10.9	4.9	15.8	17.0	7.6	24.6
XIII	6.2	5.7	12.1	11.1	10.0	21.1	17.3	15.7	33.0
XIV	6.5	2.4	8.9	11.5	4.3	15.8	17.8	6.8	24.6

Table 3 gives coefficients for the loss of head through the water column in terms of the velocity head $(\frac{v^2}{2g})$. If n represents this coefficient, the loss of head in feet equals $n\frac{v^2}{2g}$, v being the velocity in feet per second and g the acceleration of gravity. The nominal size of the column was used in calculating the velocity. The coefficients give a good means of comparing the losses in the

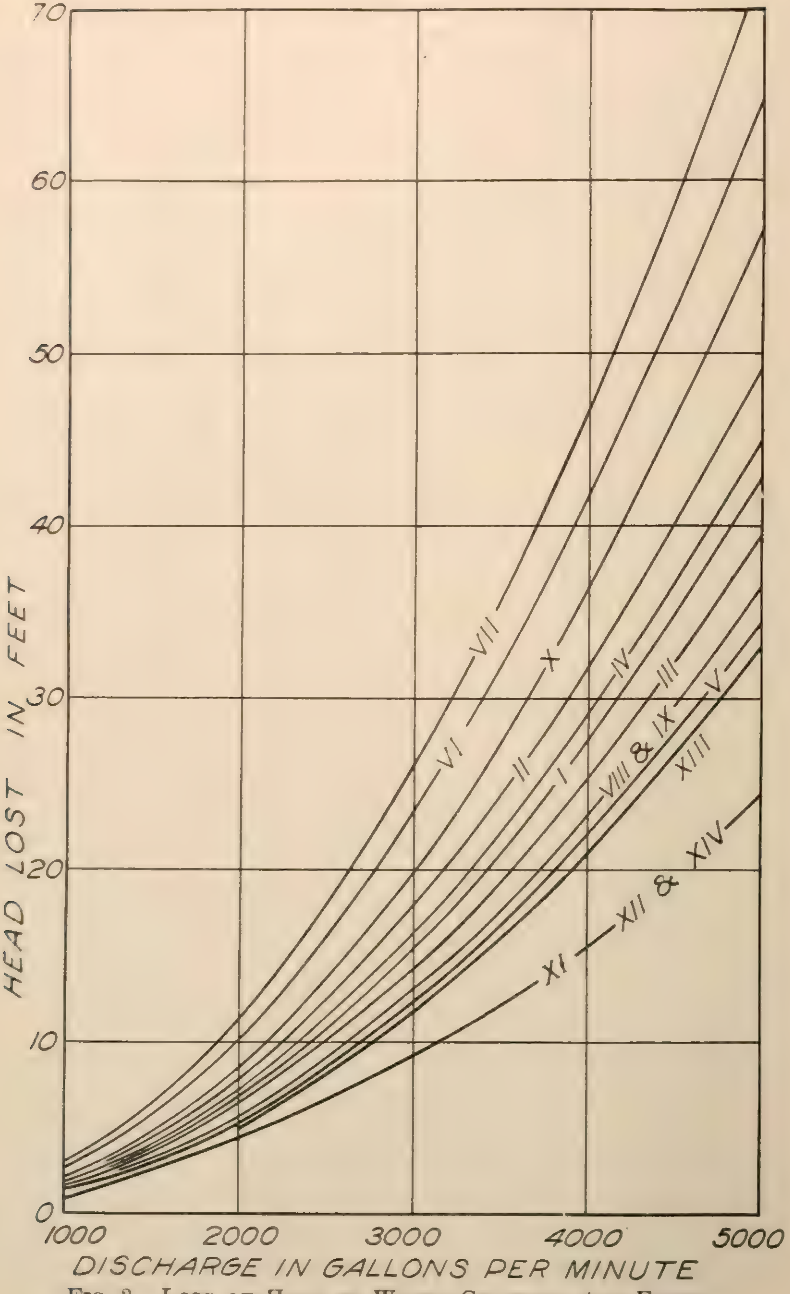

FIG. 2. LOSS OF HEAD IN WATER COLUMNS, ALL FORMS.

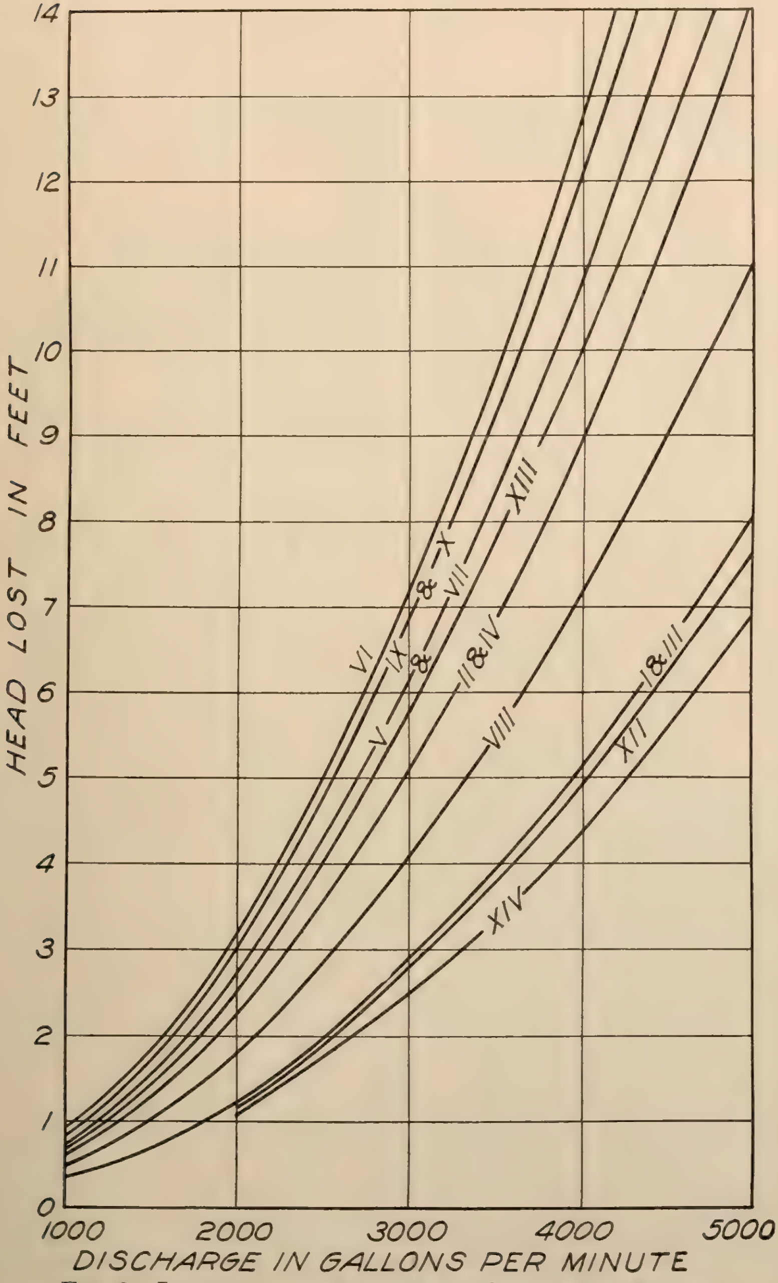

FIG 3. LOSS OF HEAD IN RISER AND SPOUT, ALL FORMS.

water columns with the losses by friction in supply pipe, bends, tees, etc. The table also gives the equivalent length of new cast-iron pipe which will give the same frictional loss as is found in the water column itself. As this length will vary somewhat with the velocity of flow, the tabular values are based on a discharge of 4000 gallons per minute.

TABLE 3.

Coefficient of Loss of Head in 10-in. Water Columns and the Equivalent Length of Straight Pipe giving the Same Friction Loss.

Designation	Coefficient n in $n\frac{v^2}{2g}$	Equivalent Length of Pipe		
		10-in.	12-in.	14-in.
I	6.72	338	791	1,690
II	7.65	384	900	1,920
III	6.14	308	722	1,540
IV	7.04	354	829	1,770
V	5.41	272	637	1,360
VI	10.20	510	1,195	2,550
VII	11.30	566	1,330	2,840
VIII	5.58	280	800	1.400
IX	5.65	284	811	1,420
X	8.83	444	1,040	2.220
XI	3.60	186	437	940
XII	3.67	190	446	960
XIII	4.94	256	603	1,295
XIV	3.67	190	446	960

TABLE 4.

Loss of Head in 10-in. Water Columns.

The velocities are given in feet per second and the loss of head in feet. These values may be considered also to be applicable to 8-in. and 12-in. water columns. The loss given does not include the velocity head of the entering water.

Designation	Loss of Head for Velocity Given in Caption			
	10	12	15	20
I	10.3	14.6	23.2	41.4
II	11.7	16.9	26.4	47.0
III	9.4	13.5	21.2	37.6
IV	10.8	15.5	24.2	43.1
V	8.3	12.0	18.7	33.5
VI	15.6	22.5	35.1	62.4
VII	17.3	25.0	39.0	69.3
VIII	8.6	12.3	19.3	34.3
IX	8.7	12.5	19.6	34.8
X	13.5	19.5	30.4	54.0
XI	5.6	8.0	12.6	22.4
XII	5.7	8.2	12.9	22.8
XIII	7.7	11.1	17.3	30.7
XIV	5.7	8.2	12.9	22.8

Table 4 gives the loss of head in 10-in. water columns for four velocities. It may be considered to be applicable within fair limits of accuracy to 8-in. and 12 in. water columns.

Table 5 gives losses for given discharges through 8-in., 10-in., and 12-in. water columns. These tables will be convenient for comparison.

TABLE 5.

Loss of Head in Water Columns.

This table is based on the loss found in 10-in. columns and on the assumption that the loss of head at any given velocity of water is the same for the different sizes of columns of a given type. The loss is given in feet of head of water and the discharge in gallons per minute.

Designation	8-in.		10-in.		12-in.	
	2,000	2,500	3,000	4,000	4,500	6,000
I	16.7	26.0	15.5	27.7	16.7	30.0
II	19.0	30.0	17.7	31.5	19.0	24.0
III	15.6	24.6	14.2	25.3	15.6	28.2
IV	17.3	27.0	16.2	29.0	17.3	31.0
V	13.5	21.0	12.5	22.3	13.5	24.0
VI	25.1	39.5	23.5	41.8	25.1	45.0
VII	28.0	43.0	26.1	46.5	28.0	50.0
VIII	14.0	22.0	13.0	23.0	14.0	25.0
IX	14.1	22.2	13.1	23.3	14.1	25.5
X	22.0	24.5	20.3	36.4	22.0	39.5
XI	9.5	14.6	8.7	15.4	9.2	16.7
XII	9.7	14.9	8.9	15.8	9.6	17.0
XIII	13.2	20.5	12.1	21.1	13.0	22.4
XIV	9.6	14.9	8.9	15.8	9.6	17.0

10. *Comments on Loss of Head.*—By examining the water columns it may be seen that the form of some of them is not in accordance with the principles of good hydraulic design. Sudden changes in direction, sudden contraction and expansion of the section of the stream, and tortuous passages are among the objections found, and high local velocities are especially troublesome. The high losses found in some of the water columns were, therefore, not unexpected. It seems probable that the need of providing certain mechanical features was placed uppermost in making the design, and that the hydraulic requirements were neglected. Some of the columns which have excellent mechanical features give high hydraulic losses. It goes without saying that both hydraulic and mechanical features are important matters in the design of a water column. As an illustration of modifications in design which will give improved hydraulic conditions without loss

in mechanical advantages, water columns XII and XIII may be cited, as compared with water columns III and IV.

It is evident that the rate of discharge of water columns under working conditions is much less than many engineers have supposed and that water columns are frequently rated too high in the catalogs of manufacturers. To put it the other way, the loss of head for a given discharge is considerably higher than it has been estimated to be, and probably higher than is necessary. A comparison with the length of pipe which will give a friction loss equal to the loss through a 10-in. water column may help to give an appreciation of the great loss of head in the water column. It may be shown that at a given discharge the loss through a 10-in. water column is equal to the frictional loss for the same discharge through a 10-in. pipe line 186 ft. long with the water column having the lowest loss of any in the series except XI, and 566 ft. long with the one having highest loss; or if a 12-in. pipe be used to feed the 10-in. water column a line 437 ft. long would give the same friction loss as the first-named water column, and one 1330 ft. long the same as the last-named water column. If it is necessary to lift the water to a greater elevation to provide the necessary head, it is evident that the cost of pumping water to the extra height and also the expense of giving a greater elevation to the tanks must be considerable. Or, to view it from another point, the rate of discharge for ordinary elevations of supply tank will be much smaller than is generally expected, and smaller than may be obtained with more efficient water columns. The use of larger water columns suggests itself as one way of keeping the lost head to a reasonable limit, though it is reported that firemen have some difficulty in handling some types of adjustable spouts of the larger columns.

Another indication of the large loss through water columns is seen in the large values of the coefficient n given in Table 3. A comparison with the coefficients for the loss of head in bends and valves brings this out in a striking manner.

11. *The Discharging Capacity of Water Columns.*—Since the head available in a water column depends upon local and other conditions, and the economic problem is so varied, the selection of a limiting or maximum velocity of flow upon which general design may be based can not be made. However, it is evident that the economical velocity through a water column and also through

the supply main will be much above the limiting velocity used in ordinary water-works practice where continuous flow throughout twenty-four hours and the cost of pumping against the friction head of long lines of mains combine to make the economical velocity less than 5 ft. per second and sometimes as low as 3 ft. per second. Aside from such considerations as the cost of pumping and that of giving sufficient elevation to the supply tanks, the maximum velocity allowable through a water column will depend upon such matters as the satisfactory operation of valve and the effect of closing the valve in producing water hammer in the supply main. From a study of these tests and a comparison with water-works practice, it would seem that with a short line of pipe from the supply tank a velocity of 12 or 15 ft. per second through the water column may be considered as the maximum desirable velocity for ordinary conditions, and for longer lines the limiting velocity should be smaller. For a long line of supply main the limit of allowable velocity would be perhaps as low as 8 ft. per second. It would seem, then, that 3000 gal. per min. for an 8-in. water column, 4000 gal. per min. for a 10-in. water column, and 6000 gal. per min. for a 12-in. water column may perhaps be considered to be the limit of desirable flow through water columns. It would also appear that a loss of much more than 20 ft. of head for the discharges just mentioned may be considered to be excessive, under conditions of ordinary tank supply.

12. *Calculations for Water Service Installation.*—Although the calculations involved in the design of sizes for water service plants are not within the scope of this bulletin, it may be convenient to some readers to include here a brief review of a method which may be used in making calculations of sizes of pipe mains and loss of head. The method here given follows the report of the Committee on Water Service of the American Railway Engineering and Maintenance of Way Association*, the method being the joint work of Mr. Ferriday and the writers.

The total head producing flow (called by the Committee the "flow head") is utilized in overcoming resistances and is equivalent to the sum of several items,—velocity head of the flowing water, loss at entrance, friction in pipe, loss in elbows, etc., and loss through the water column. This may be expressed by the following equation:

---

* See the Proceedings of the American Railway Engineering and Maintenance of Way Association, vol. 11, p. 1161.

$$H = m\frac{v^2}{2g} + e\frac{v^2}{2g} + h_1 + n\frac{v^2}{2g} + \frac{v^2}{2g} \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (1)$$

where H is the "flow head", $m\frac{v^2}{2v}$, $e\frac{v^2}{2g}$ and $n\frac{v^2}{2g}$ are the losses in head at entrance, in elbows and tees, and in the water column, respectively, $\frac{v^2}{2g}$ is the velocity head of the issuing water, and h_1 is the frictional loss in the pipe line. The values h_1 and the losses in elbows and tees may be obtained from the diagrams in Fig. 11, 12, and 13, which are explained in the Appendix. For these diagrams the units are feet and seconds. When the supply main and the water column are not of the same diameter, the v to be used in the several terms will be that for the size in which the loss occurs.

If it is desired to know the flow head necessary to produce a given discharge, substitute values in equation (1). Thus, if 6000 gal. per min. is to be discharged through 200 ft. of 14 in. pipe and there is one elbow with a radius of curvature equal to 1½ diameters and if the water column used is a 12-in., Type V, the following values of the heads will be obtained, if we neglect loss in contraction of section from 14-in. pipe to 12-in. column:

	Feet of head.
Entrance head $m\frac{v^2}{2g}$ (Fig. 13, $m = 1$)	2.5
Loss in one elbow $e\frac{v^2}{2g}$ (Fig. 13)	1.1
Loss in 200 ft. of 14-in. pipe, (Fig. 12)	6.6
Loss in water column (Fig. 23)	24.0
Velocity head $\frac{v^2}{2g}$ (Fig. 13)	4.5
	38.7

That is, it will require that the level of water in the supply tank remain not less than 38.7 ft. above the level of the discharge point of the water column.

If a given combination of pipe line and water column and a specified flow head be the known quantities and it is desired to find the rate of discharge, the most convenient process is a trial method. Assume a discharge as close to the probable discharge as may be estimated and by equation (1) calculate the flow head required to give this discharge with the combination of pipe line and water column. Divide the specified flow head by the flow head thus found by calculation and call the quotient the trial ratio. Multiply the water column loss used in the calculation by this ratio; the product may be considered an approximation to the loss which will be found in the water column under the conditions specified. From the diagram for water column losses find the discharge which corresponds to this water column loss. The amount so found will be approximately the discharge of the given combination of pipe line and water column. As a check, the head required to give this discharge may be computed by equation (1), and if the result is not close enough the steps indicated may be gone through again, using as a new trial ratio the quotient of the specified flow head by the flow head obtained by this calculation. The result of the first calculation will usually be within the required degree of accuracy.

As an example, let it be required to find the discharge through 400 ft. of 12-in. pipe and a 10-in. water column of Type VIII, the line having two elbows with radius of curvature of $1\frac{1}{2}$ diameters and the controlling level of the water in the tank being 30 ft. above the discharge orifice of the water column. Assume that the discharge will be 3000 gal. per minute. Then the necessary head by equation (1) may be found as follows:

	Feet of head
Entrance head, $m\frac{v^2}{2g}$ (Fig. 13, $m = 1$)	1.1
Loss in two elbows, $e\frac{v^2}{2g}$ (Fig. 13)	1.0
Loss in 400 ft. of 12-in. pipe (Fig. 12)	8.4
Loss in water column (Fig. 29)	13.0
Velocity head (10-in.) (Fig. 13)	2.5
Total head required,	26.0

The trial ratio then is $\frac{30}{26} = 1.15$, and the loss through the water column may be expected to be $13.0 \times 1.15 = 15.0$ ft. By Fig. 29, the discharge through the water column for a loss of head of 15 ft. is 3250 gal. per min. Using equation (1) for a discharge of

3250 gal. per min., the calculated loss is found to be 29.9 ft., so that the discharge of 3250 gal. per min. may be considered to be sufficiently close.

Table 6 gives examples of combinations of pipe lines and water columns under assumed conditions of height of tank and flow head, length of pipe line, and kind of water column, with summarized results of calculations of the discharge and the distribution of the flow head among pipe friction, loss in water column, etc.

TABLE 6.

EXAMPLES SHOWING THE DISCHARGE AND THE LOSSES FOR VARIOUS COMBINATIONS OF PIPE LINES AND WATER COLUMNS.

Distances, heads and losses are given in feet; diameters in inches; discharges in gallons per minute.

Depth in Tank	Tank Bottom above Rail.	Pipe Line		Water Column		Discharge	Entrance Loss	Velocity Head	Elbow Loss	Pipe Friction	Column Loss	Total
		Length	Diam	Dia.	Des.							
16.0	20.0	1000	12	10	V	2 370	0.7	1.5	0.5	13.5	7.8	24.0
5.0	20.0	1000	12	10	V	1 700	0.3	0.7	0.4	7.6	4.0	13.0
16.0	20.0	1000	14	10	V	2 880	0.6	2.3	0.5	9.1	11.5	24.0
5.0	20.0	1000	14	10	V	2 110	0.4	1.1	0.3	5.1	6.2	13.0
16.0	20.0	1000	12	12	V	2 690	0.8	0.8	0.7	17.0	4.7	24.0
16.0	20.0	1000	14	12	V	3 500	0.9	1.5	0.8	12.8	8.0	24.0
16.0	20.0	1000	16	12	V	3 950	0.6	1.8	0.6	8.7	10.4	24.0
5.0	20.0	1000	16	12	V	2 950	0.4	1.2	0.3	5.2	5.9	13.0
16.0	20.0	1000	14	10	XII	3 070	0.8	2.5	0.5	10.0	9.3	23.2
16.0	20.0	1000	14	10	VI	2 410	0.5	1.6	0.4	6.5	15.0	24.0
16.0	20.0	500	10	10	XII	2 250	1.4	1.4	1.3	14.1	5.0	23.2
16.0	20.0	500	12	10	XII	2 950	1.1	2.3	0.8	10.3	8.7	23.2
16.0	20.0	500	14	10	XII	3 490	0.7	2.8	0.7	6.4	12.0	23.2
16.0	20.0	500	14	10	VI	2 580	0.5	1.9	0.5	3.7	17.4	24.0
16.0	20.0	200	10	10	XII	2 830	2.2	2.2	2.0	8.8	8.0	23.2
16.0	20.0	200	12	10	XII	3 450	1.4	2.4	1.3	5.4	11.7	23.2
16.0	20.0	200	14	10	XII	3 800	1.0	3.8	0.9	3.0	14.4	23.2
16.0	20.0	200	14	10	VI	2 720	0.6	2.0	0.5	1.7	19.2	24.0
16.0	20.0	100	10	10	XII	3 160	2.4	2.7	2.8	5.4	9.9	23.2
16.0	20.0	100	12	10	XII	3 690	1.8	3.5	1.5	3.0	13.4	23.2
16.0	20.0	100	14	10	XII	3 910	1.1	4.1	1.1	1.6	15.3	23.2
16.0	32.0	1000	14	10	V	3 560	1.0	3.4	0.8	13.2	17.6	36.0
16.0	32.0	500	14	10	XII	4 300	1.4	5.0	1.1	9.3	18.4	35.2
16.0	32.0	200	12	10	XII	4 280	2.3	4.8	2.1	7.8	18.2	35.2
16.0	32.0	100	12	10	XII	4 520	2.7	5.4	2.3	4.4	20.4	35.2
16.0	32.0	100	14	10	XII	4 870	1.6	6.2	1.5	2.3	23.6	35.2
16.0	32.0	100	14	10	VII	3 160	0.8	2.7	2.4	1.1	29.0	36.0
16.0	32.0	100	14	12	XII	6 500	2.9	5.5	2.6	4.9	20.2	35.2

13. *Principles of Water Hammer Caused by Sudden Stoppage of Flow.**—When a valve at the end of the pipe line is suddenly closed, additional water pressure is produced throughout the pipe line. Some of the conditions and results attending the phenomena may be briefly stated as follows:

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* A bulletin of the Engineering Experiment Station on Water Hammer in Pipes is in preparation and will treat more fully the matters referred to in this article.

(1) The energy of the moving water is expended in compressing the water in the pipe and in expanding the walls of the pipe.

(2) An increased pressure is developed within the pipe, and the increase of pressure is called water hammer.

(3) The water next to the valve is immediately stopped and the full amount of water hammer is developed instantly at that point, the term "instantly" being here used to mean a very short time.

(4) The amount of this water hammer in pounds per square inch in pipes of the size and kind here considered is about fifty-four times as much as the original velocity of the water in feet per second.

(5) The water in the pipe away from the valve is stopped and compressed layer by layer, the full amount of the water hammer developing almost instantly at any point as soon as the water between that point and the valve has received its compression.

(6) This pressure wave or impulse of the water hammer (and the time when each layer receives its compression) moves along the pipe line away from the valve with a speed of about 4000 ft. per sec. for the size of pipe ordinarily used in supply mains. The rate of travel is the same as the velocity of sound in water as modified by the thickness and elasticity of the walls of the pipe.

(7) When a pressure wave has traveled to a free water surface or to a larger body of water, a release of pressure begins at this relief point and travels backward at the same speed, about 4000 ft. per sec. The water at any point in the pipe is held at its full water hammer pressure while this impulse or pressure wave travels on to the point of release and back. At the valve the pressure will last the longest. Thus for a pipe 2000 ft. long, the water hammer pressure at the valve will be maintained for one second.

(8) This first pressure or first pulsation of pressure is followed by a series of pulsations of diminishing intensity. The water in the pipe may be considered as a springy substance bounding and rebounding from the closed end.

(9) The pressure developed when closure is sudden, is expressed by the formula, the derivation of which will not be given here:

$$P = \frac{1}{\sqrt{1 + \frac{Ed}{E't}}} \sqrt{\frac{wE}{144g}}\, v \text{..........................} (2).$$

in which

$P =$ water hammer pressure (lb. per sq. in.), the maximum additional pressure for a given velocity of flow.

$v =$ the velocity of the water in the pipe before valve was closed (ft. per sec.)

$E =$ modulus of elasticity of the water (300 000 lb. per sq. in.)

$E' =$ modulus of elasticity of the metal in pipe (30 000-000 lb. per sq. in. for steel and 15 000 000 lb. per sq. in. for cast iron.)

$d =$ diameter of pipe.

$t =$ thickness of pipe wall.

$g =$ acceleration of gravity (32.2 ft. per sec. per sec.)

$w =$ weight of a cubic unit of water (62.5 lb. per cu. ft.)

For cast-iron pipe of the sizes used as supply lines the formula reduces to approximately

$$P = 54\, v \text{..............................} (3).$$

(10) By sudden closing of the valve is meant that the effective part of the closure takes place in less than the time required for the first impulse to travel to the tank and back to the valve at a rate of approximately 4000 ft. per sec. This velocity (V) of the pressure wave is given by the formula

$$V = \frac{4710}{\sqrt{1 + \frac{Ed}{E't}}} \text{..................................} (4).$$

The effective part of the closure begins when the resistance through the valve opening becomes large enough to give a considerable check to the flow of water. For the water columns under consideration, the effective part of the closure may be said to be the last 15 % to 25 % of the valve movement.

(11) The length of the pipe line does not enter into the expression for the amount of water hammer developed by sudden closure of the valve. It does affect the length of the time the water is under compression and hence in a way the damage which may be done. The length of pipe line is of importance in the problem in that if the valve is closed in a shorter length of time than is necessary for the wave to travel out and back, the resulting water hammer is the same as that which would be obtained with instantaneous closing of the valve. The pressure produced when the valve closure is slow is discussed in a later paragraph.

14. *The Effect of Valve Movement Upon Pressure.*—The following theoretical investigation will assist in obtaining an idea of what is taking place in the supply line after the valve is closed, first for the case of slow closure, and second for the case of sudden closure. In a long line of pipe when steady flow has become established,

$$H = m\frac{v^2}{2g} + f\frac{l}{d}\frac{v^2}{2g} + e\frac{v^2}{2g} + n\frac{v^2}{2g} + q\frac{v^2}{2g} + \frac{v^2}{2g} \quad \ldots\ldots\ldots\ldots (5)$$

$$= [\,(m + f\frac{l}{d} + e + n + 1) + q]\,\frac{v^2}{2g} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (6)$$

in which q is the coefficient of valve loss, a variable whose value depends upon the amount the valve is closed; $m\frac{v^2}{2g}$ is entrance loss, $f\frac{l}{d}\frac{v^2}{2g}$ the loss in pipe friction, $e\frac{v^2}{2g}$ the loss in elbows, $n\frac{v^2}{2g}$ the column loss and $\frac{v^2}{2g}$ the velocity head. For simplicity place

$$m + f\frac{l}{d} + e + n + 1 = a$$

Then

$$H = (a + q)\frac{v^2}{2g}$$

and

$$v = \sqrt{\frac{2gH}{a + q}} \quad \ldots\ldots\ldots\ldots\ldots\ldots (7)$$

It will be noticed that the velocity of flow produced by the head H depends upon both a and q. As an illustration, it has been found that for a gate valve 82 % closed $q = 41$. Then if

$a =$ 13.7, the velocity in the pipe after steady flow has become established will be $\frac{1}{2}$ of the velocity in the pipe when the valve is open; if

$a =$ 53.0, the velocity will be $\frac{3}{4}$ of that with valve open; if

$a =$ 175.6, the velocity will be $\frac{9}{10}$ of that with valve open.

These values show that the velocity of flow is affected but little in a long pipe during the first 80 % of the closing of the valve, the three values of a corresponding to lengths of 12-in. pipe of 685 ft., 2660 ft., and 8800 ft., respectively. In Fig. 4 have been plotted the results of calculations with equation (7), using Weisbach's values of q for gate valves and five values of a. The curves show how the velocity in the pipe diminishes as a very slow closure of valve is made, the total head remaining constant. To use the diagram, the value of a for the pipe line should be computed. As an illustration, for a 12-inch pipe line 500 ft. long, with four elbows and with a 10-inch water column, a equals 12.4, and the values may be interpolated between the lines $a = 10$ and $a = 20$. It should

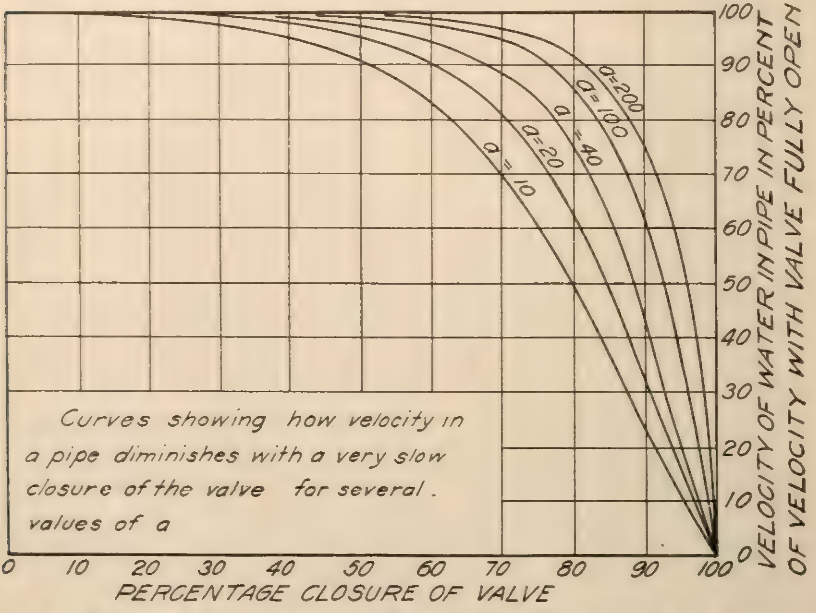

FIG. 4. DIAGRAM SHOWING THE VELOCITY IN PIPE LINES OF VARIOUS LENGTHS FOR ANY VALVE POSITION,—STEADY FLOW CONDITIONS.

be noted that a should be computed using a velocity head based upon a diameter of pipe equal to the size of the pipe line.

When the valve is closed quickly, equation (7) does not apply because a water hammer pressure is produced which forces the water past the valve and hence keeps the velocity in the pipe at a higher value than it would have during a very gradual closure. If the valve is closed in less time than $\frac{2l}{4000}$ seconds, it may be considered a sudden closure, and the velocity in the pipe near the valve at any valve position can be computed as follows. During a partial closure, the value of the valve coefficient q has changed from q_1 to q_2, and the head or resistance at the valve (neglecting the back pressure from the water column itself) is now

$$P = q_2 \frac{v^2}{2g} \qquad \text{(8)}$$

Call the velocity v_s after the flow has become steady. The pressure at the valve at that time becomes

$$P_s = q_2 \frac{v_s^2}{2g} \qquad \text{(9)}$$

v_s can be computed from equation (7) when q_2 is known, whatever the position of the valve.

From equations (8) and (9)

$$v = v_s \sqrt{\frac{P}{P_s}} \qquad \text{(10)}$$

If the velocity was originally v_o the water hammer generated would be

$$P = 54\,(v_o - v) \qquad \text{(11)}$$

Substituting for P in equation (10),

$$v^2 = \frac{54\, v_s^2}{P_s}\,(v_o - v)$$

or

$$v = -\frac{54\, v_s^2}{P_s} + \sqrt{\frac{54\, v_s^2 v_o}{P_s} + \left[\frac{54 v_s^2}{P_s}\right]^2} \qquad \text{(12)}$$

Knowing v, P can be found from equation (8).

If the values of q are known for any valve, the manner of increase of pressure during a rapid closing of the valve can be easily determined. In Fig. 5 have been plotted the results of calculations with equations (8) and (12) using Weisbach's values of q for gate valves. The curve shows the way in which the velocity

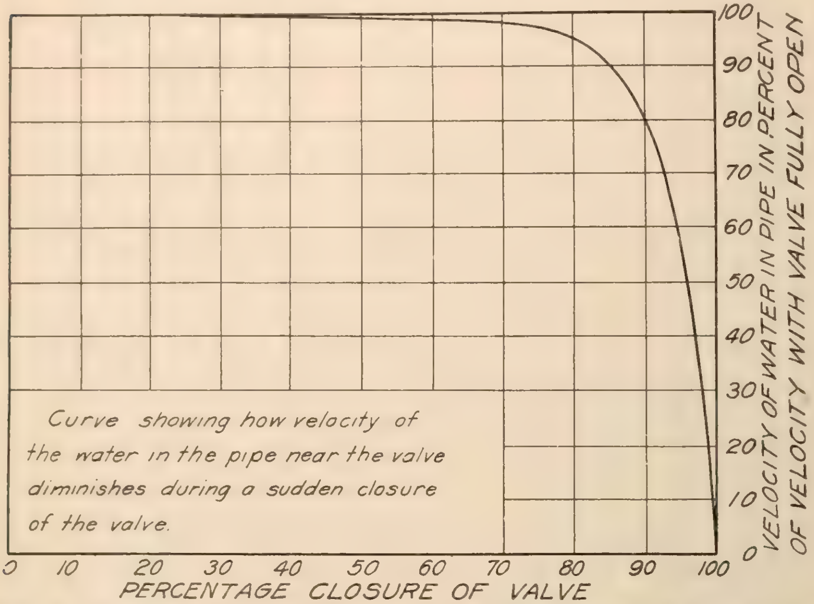

FIG. 5. DIAGRAM SHOWING VELOCITY IN A PIPE LINE FOR ANY VALVE POSITION DURING A SUDDEN CLOSURE OF VALVE.

of the water in the pipe near the valve diminishes during a sudden closure of the valve. Note that the velocity is 80 per cent of its original value when valve is 90 per cent closed.

In Fig. 6 are curves worked out with equations (7) and (8) for pipe of various lengths. These curves show the amount of pressure or resistance developed during a very slow closure of a gate valve for different values of a. Note that for very long pipe lines, the pressure against the valve (resistance of valve opening) does not become any considerable proportion of the total flow head until the valve is more than 90 per cent closed. In short pipe lines, closure of the valve is felt much earlier, as is seen in the curves for the smaller values of a.

Fig. 7 shows how the pressure would rise in a pipe of any length if the valve is closed suddenly. The length of pipe need not be considered because it does not enter into equations (8) and (12).

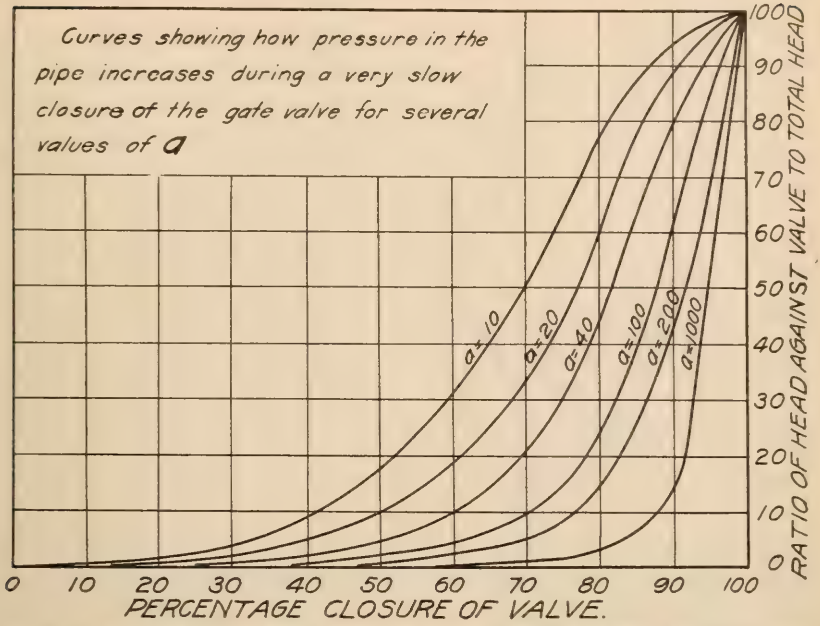

FIG. 6. DIAGRAM SHOWING PRESSURE ON GATE VALVE AT END OF PIPE LINE FOR ANY VALVE POSITION, AND FOR VARIOUS LENGTHS OF PIPE—STEADY FLOW CONDITIONS.

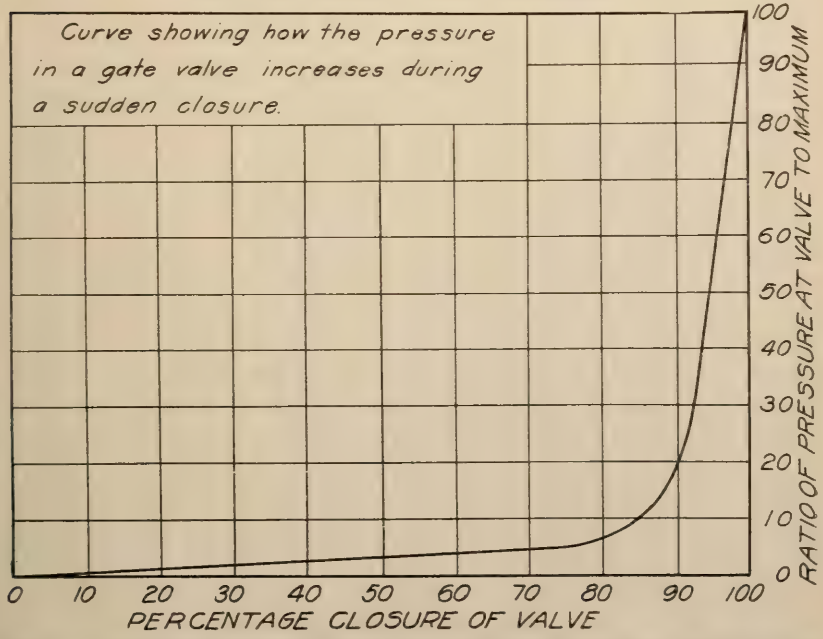

FIG. 7. DIAGRAM SHOWING PRESSURE ON GATE VALVE AT END OF PIPE LINE FOR ANY VALVE POSITION DURING A SUDDEN CLOSURE.

15. *Time-pressure and Valve-movement-pressure, Diagrams.*—The valve A (Fig. 1, page 6) was partially closed, so as to give added resistance to flow in the pipe line. As valve E closes it consumes more and more of the pressure until the entire static pressure is taken. Observations were first made with valve A slightly closed. The valve A was then closed somewhat more and another experiment made. The pressure at C was recorded by a Crosby indicator in two ways: (1) on a drum rotating at a uniform rate, (2) with the drum connected by a cord to the valve mechanism and moving as the valve moved. An electric attachment recorded half seconds on both forms of diagram. Fig. 8 and 9 (page 28 and 29) give the results of the tests made in this way. The diagrams are here called (1) time-pressure diagrams and (2) valve-movement-pressure diagrams, respectively.

Under the conditions of flow

$$H = b\frac{v^2}{2g} + q\frac{v^2}{2g}$$

where $b\frac{v^2}{2g}$ represents all losses except that due to the water column valve. Hence

$$v = \sqrt{\frac{2gH}{b+q}} \quad \text{.............................} \quad (13)$$

which is similar to equation (7).

By means of the valve A any value of *b* can be obtained, and as *b* is similar to *a* in equation (7), a diagram can be produced which will represent the manner in which pressure rises in any pipe line. The valve-movement-pressure diagrams are similar to the curves in Fig. 6 (page 25). It will be noted from Fig. 6 that there is not much change in the shape of the pressure curve when *a* is greater than, say, 40. It was not possible to obtain this form of diagram from some of the hydraulically operated valves because the moving parts are not accessible. It will be noticed on diagrams of water columns I, II, III, IV, V, and XIV, shown in Fig. 9, that the rise does not begin until the valve is nearly 95 per cent closed. From Fig. 6 it is seen that with a gate valve in a pipe line which has *a* even as high as 1000 the pressure begins to rise at 90 per cent closure. The difference can be explained by the construction of the valves. A cylinder valve closes off a much larger fraction of total area in the last 10 per cent of the closure than does a gate valve. In column VII it is seen that the pressure begins to rise at 40 per cent of the closure. This was caused, at least in part, by the very short valve travel. The travel in this case was so short that some of the great loss of head for this column must be attributed to the smallness of the opening.

The time-pressure diagrams were obtained for all the water columns and are shown in Fig. 8. A study of these shows that except in water columns XII and XIII a very short time is used for what may be called the effective portion of closure—that is the portion of the closure which produces the principal retardation of the water in the pipe. Columns I and II use 17 seconds for closing, and the effective portion of closure occupies only a little more than one second. With a pipe line 2000 feet long or more, great water hammer pressures would be produced unless prevented by a relief valve. Columns XII and XIII have the first portion of the closure made very rapidly, and the last or the effective portion of the closure is made very slowly. It may be seen from Fig. 9 that in one experiment with columns XII and XIII the first 83% of the closure was made in 2 sec., and the rest of the closure was made in 11 sec. In its effect upon water hammer this is the same as closing the valve at a uniform rate in 65 sec. instead of in the 13 sec. which was actually used; columns III, IV, V, IX, and X make the effective portion of closure in about $\frac{1}{2}$ second. A supply line 1000 feet long would be liable to have destructive water hammer unless protected by relief valves. Columns VI and VIII are hydraulically operated, and both show that the closure is hastened by an increased pressure in the supply line. Column VI has 3 seconds for the shortest effective portion of closure shown on diagram. Triangular ports are the reason in this case for the comparatively long time of effective closure. The valve must move a relatively large amount near the end of the closure to cover up a given area of port. A disadvantage of the triangular port may be seen by a comparison of the loss of head with column X which is similar in almost every respect but has rectangular ports. At 4000 gal. per min. 29 feet of head are lost in the valve of column VI against 24.4 feet in valve of column X.

16. *Application to Formation of Water Hammer in Supply Lines.*—The supply line used in the experiments was too short to permit water hammer to be developed to any extent. The time-

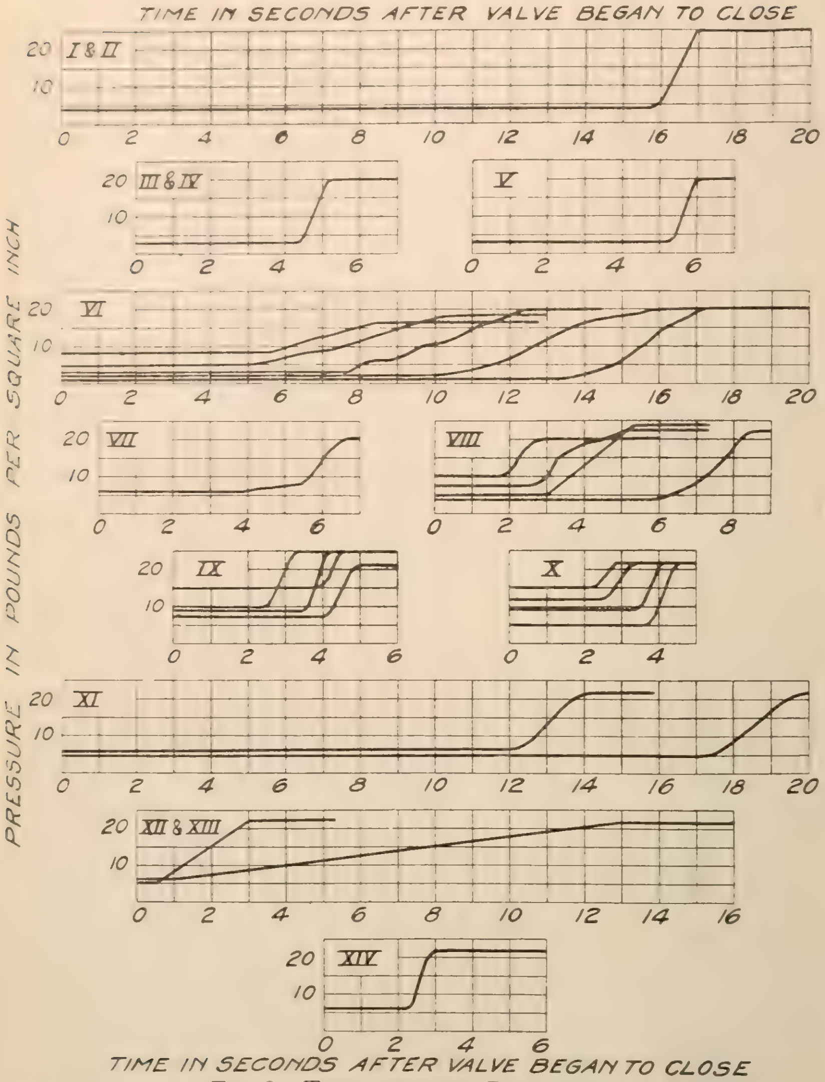

FIG. 8. TIME-PRESSURE DIAGRAMS.

pressure and valve-movement-pressure diagrams give information from which the formation of water hammer in supply lines may be forecast.

(1). Velocities as high as 15 ft. per sec. in the supply line of a water column are not uncommon, while a velocity as high as 10 ft. per sec. is quite usual. With such large quantities of water flowing it is necessary that the water column be shut off promptly when the locomotive tender is full. The time required in closing the valves experimented upon ranged from 3 to 17 seconds, most of the valves closing in less than 8 seconds. The high velocity and the short time of closure make destructive water hammer possible unless means are used to prevent it. With a velocity of 10 ft. per sec. in the supply line, 540 lb. per sq. in. water hammer is possible.

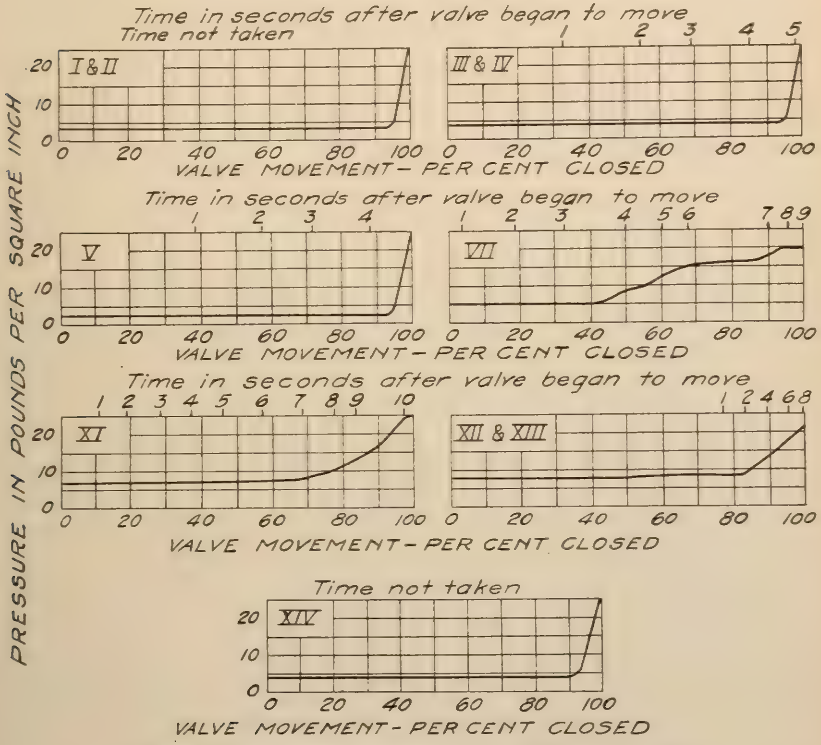

FIG. 9. VALVE-MOVEMENT-PRESSURE DIAGRAMS.

(2). From an examination of the time-pressure and valve-movement-pressure diagrams (Fig. 8 and 9) it will be noted in almost all the makes that the pressure does not begin to rise rapidly until the valve is about ninety per cent closed and until only a very short time remains before valve is completely closed. Water column VII (Mansfield) is an exception. The remaining part of the closure is the effective portion of closure.

(3). If the time in seconds taken by the effective portion of the closure is equal to or less than twice the length of the supply line in feet divided by 4000, maximum water hammer will be caused. If there is no relief valve or other device for reducing the water hammer, the pressure in pounds per square inch will be approximately 54 times the velocity of the water in feet per second in the supply line before shutting valve.

(4). If the time in seconds taken by the effective portion of the closure is greater than twice the length of the supply line in feet divided by 4000, maximum water hammer will not be caused. The magnitude of the pressure will depend upon the manner in which the last part of the closure is made.

(5). It is immaterial how quickly the first eighty per cent or so of the closure is made. By making this part of the closure in less time, the time saved can be used in making the last part of the closure more gradual. For example, assume that a valve is closed at a uniform rate in 10 sec. Reducing the time of making eighty per cent of the closure to 4 sec. and making the last twenty per cent of the closure in 6 sec. keep the total time of closure at 10 sec., but its effect on water hammer is the same as if a uniform closure were made in 30 sec.

This statement which was formulated before the later tests were made is well exemplified in an experiment with water column XII in which the first 83% of closure was made in 2 sec., and the last 17% of closure in 11 sec. (See Fig. 8 and 9, page 28 and 29.)

17. *Pressures Produced by Slow Closure of Valve.*—In some of the preceding paragraphs the water hammer produced by sudden closure of valve was discussed. When the time taken to make the effective part of the valve closure is greater than the time required for the pressure wave to make a circuit out and back, the amount of water hammer will be less than given by the formula for maximum water hammer. In case the time taken for closure becomes so much greater that the pressure at the valve is hardly more than the resistance which the partially closed valve gives to the passage of the amount of water carried at a given instant (and hence no real water hammer is produced or so little that it need not be considered), the problem is of a quite different nature. The following method for determining the effect of slow closure may be used.

From the principle of impulse and momentum,

$$\int_{t'}^{t} P dt = \int_{v'}^{v} \frac{62.5\ l}{32.2 \text{x} 144} dv \dots\dots\dots\dots\dots (14)$$

where

P = resultant pressure in lb. per sq. in. at any instant.
t = time in seconds.
v = velocity in pipe in ft. per sec. at any instant.
l = length of pipe in feet.

For very short intervals of time and small changes of velocity, the approximate form may be used:

$$P(t-t') = 0.0135\ l\ (v-v') \dots\dots\dots\dots\dots (15)$$

If the law of the valve movement and the corresponding valve resistance during closure are known, (that is, if the time-pressure diagram and the valve-movement-pressure are given, or if the coefficients of valve resistance and the law of valve closure are known), the pressures produced during slow closure may be calculated approximately by the use of equations (8) and (15). As an illustration, the additional pressures developed in a 10-in. pipe line 1000 ft. long with flow head of 45 ft. have been calculated, by an equation not given here, for a cylindrical valve for various times of closing at a uniform rate and plotted in Fig. 10. If the valve does not close at a uniform rate throughout and if the rate of closure through the portion which gives fairly high valve resistance may be considered as uniform (say the last 15% of valve movement), the diagram of Fig. 10 may be made to apply fairly closely by dividing the time of making the effective portion of the closure by 0.15 (or whatever the proportional closure is for the effective portion of closure) and using the quotient as the full time of closure. It will be seen that considerable pressure is developed even with long-time closure. For gate valves a similar method of computation may be used.

If the resistance of the valve through the last part of closure were maintained in such a way that P would remain constant, the time required to make this part of the closure would be

$$t = \frac{0.0135\ lv}{P} \dots\dots\dots\dots\dots (16)$$

It is not practicable to obtain such a valve movement. For ordinary methods of closing, the pressure will necessarily be variable and the maximum pressure will, in general, be larger than that given by this equation. The dotted curve of Fig. 10 gives the pressures calculated by equation (16). In this diagram, the abscissas represent the full time of closing the valve. If the effective portion of closure be considered, i. e., the portion which gives relatively high resistances, the corresponding full time of closure at uniform rate should be used as the abscissa; thus, if the effective portion of closure is the last one-sixth of the valve movement, and this is made at a uniform rate in five seconds, the pressure found by equation (16) may be taken from the dotted line of the diagram by using 30 seconds as the time. Even with this interpretation, the formula is not accurate. Equation (16) has often been applied to cases of sudden closure of valve, an incorrect application unless modified by the springiness of the water and pipe.

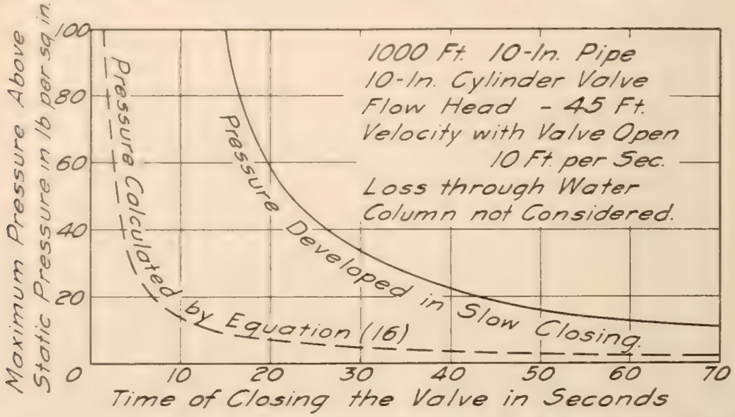

FIG. 10. CALCULATED MAXIMUM PRESSURE FOR SLOW CLOSURE OF CYLINDRICAL VALVE AT A UNIFORM RATE.

18. *Size of Relief Valve.*—It is the purpose of this article to indicate a rational method of obtaining the size of opening required for the relief valve.

Let the velocity in the supply line be represented by v. Consider that the relief valve keeps the pressure down to some value P lb. per sq. in. above static pressure, and that p is static pressure. The pressure P may be considered to be generated according to the law of water hammer. To produce this pressure P, then, the velocity in the pipe must be reduced by an amount v'. The formula for pressure produced by water hammer may be written $P = hv'$, where h is the coefficient by which the change in velocity v' is multiplied to find the pressure produced; h may be called 54 for the pipes ordinarily used for supply lines. The velocity in the pipe is now $v - \frac{P}{h}$ feet per second. The water must now all flow through the relief valve. The pressure at this point is P lb. per sq. in. above static pressure or 2.3 P feet of head. If A is area of pipe, a the area of relief valve and c the coefficient of discharge of the relief valve, then

$$A\left(v - \frac{P}{h}\right) = ca\sqrt{2g\text{x}2.3(P+p)}$$

$$a = \frac{A\left(v - \frac{P}{h}\right)}{c\sqrt{4.6g\,(P+p)}} \quad \ldots\ldots\ldots\ldots\ldots\ldots (17)$$

Substituting $h = 54$ and simplifying,

$$a = \frac{v - \frac{P}{54}}{12.2c\sqrt{P+p}}A \quad \ldots\ldots\ \ldots\ldots\ \ldots\ldots (18)$$

The value of the coefficient of discharge c of the relief valve depends upon the shape and amount of opening and upon the conditions just inside the valve; the valve may open freely into air, or the passages may be constricted at places before reaching the outlet, conditions which will cause great differences in the coefficient of discharge. The coefficient of discharge may be determined by blocking the valve open and testing its discharging capacity separate from the water column itself. The range of the coefficient, based on the nominal size, is from 0.2 to 0.6. If the outlet is constricted it will be less.

To illustrate the use of the formula, assume a long supply line in which maximum water hammer is likely to occur. Assume a velocity of 10 feet per second, and that it is desired to limit the maximum pressure to 50 pounds per square inch above the static pressure of 30 pounds per square inch. Assume $c = 0.3$.

$$a = \frac{10 - \frac{50}{54}}{12.2\text{x}0.3\sqrt{50+30}}\ A$$

$$a = 0.44\ A$$

If the supply line was 12 inch pipe, the effective area of relief valve should be $0.44 \times \frac{\pi}{4} \times 144 = 49.6$ square inches when the valve is fully open.

This method has had experimental verification.

In connection with the subject of relief valves, attention may well be called to the internal relief valve in water columns V and XIV. The relief valve forms the seat for the cylindrical valve of the water column. The location of the relief valve is a favorable one, and the size is adequate.

19. *Summary.*—The following comments on the work are offered.

1. The tests give the loss of head for various rates of discharge in the principal types of locomotive water columns in use in the United States. The loss through the valve and through the riser of the water columns were each determined.

2. At a discharge of 4000 gal. per min. the loss of head through the water columns ranged from 15.4 ft. of head to 46.5 ft. of head. The resistance to flow is surprisingly high,—much higher than has usually been estimated. It is evident that the discharge of water columns under working conditions is smaller in many cases than has been estimated. It is worth noting that the water columns which give high resistances include types which are used in large numbers by the railroads of the country.

3. A comparison with the frictional losses through pipes and elbows will help to give a fuller conception of this high loss of head. For the water column which gave the highest resistance, the loss of head is equal to the frictional loss for the same discharge through a 10-in. line of straight pipe 566 ft. long. Similarly this loss is 25 times as much as that through a 10-in. elbow. To get an adequate discharge through such a water column will require a greater elevation of tank and involve a larger expense in pumping than is necessary.

4. An examination of some of the types shows that the forms of the valves and passages do not accord with the principles of good hydraulic design. Sudden change in direction, sudden contraction and expansion of the section of the stream, and tortuous passages are among the objections to be found, and high local velocities are especially troublesome. Mechanical features of

construction, inspection, and repair seem to have crowded out consideration of hydraulic efficiency. Modifications in the design of some of the water columns, made since the first tests were conducted, have resulted in improved hydraulic conditions without loss in mechanical efficiency.

5. The telescopic adjustable spout with its large cross-section shows lower losses than the fixed spout, though it must be borne in mind that the point of discharge taken in water columns with telescopic adjustable spouts is at the end of the riser, and that for this reason the lift is greater than with the fixed spout. The ball-and-socket spout gives about the same friction loss as the rigid spout, but it has the advantage that the point of discharge is usually lower. It may also be noted that the anti-splash devices use up head in providing a solid stream.

6. The maximum velocity allowable through a water column will depend upon such matters as the satisfactory operation of the valve and also upon the effect of closing the valve in the development of water hammer in the supply main. With a short line from the supply tank, a velocity of 12 or 15 ft. per sec. through the water column may be considered as the maximum desirable velocity for ordinary conditions, and for longer lines the limiting velocity may perhaps be as low as 8 ft. per sec. It would seem that 3000 gal. per min. for an 8-in. water column, 4000 gal. per min. for a 10-in. water column, and 6000 gal. per min. for a 12-in. water column may be considered to be the limit of desirable discharge. Considerations of the cost of pipe line and supply tank required to give the above discharge in a proposed installation may show that such high velocity and discharge are uneconomical, and on the other hand it is possible that high available heads or long and expensive supply lines may make higher velocities permissible. It will be seen that the velocities named are much above the limiting velocities of ordinary waterworks practice. It would seem also that for the rates of discharge just noted the allowable loss through the water column itself should not be much more than 20 ft., and that the limit may well be placed at less than 20 ft.

7. The method outlined for making calculations of the losses of head and discharge in water service installations is a convenient one. By means of the diagrams for friction in pipe and elbows and the diagrams for loss through the water column and

the use of the trial ratio, the discharge given by any flow head may be calculated without the labor attendant upon the use of the usual formulas for flow of water in pipes.

8. The examples of combinations of pipe lines and water columns given in Table 6 illustrate the distribution of losses in such installations. There is an advantage in a large pipe by reason of the large discharge for a given head and also in the smaller opportunity for water hammer. In an installation having a high head available (as in some gravity lines) the pipe may well be smaller than in the common installation, provided the arrangement of the valve closure guards against water hammer and proper relief valves are used.

9. The water hammer pressure generated in a pipe line when a valve is closed suddenly and no relief valve is used, for the sizes and thicknesses of pipe used in ordinary water service installations, will be, in lb. per sq. in., about 54 times the velocity of water in the pipe in ft. per sec. The term "sudden closure" is here taken to mean that the time consumed in that part of the valve movement which gives relatively high valve resistance is less than the time required for the pressure impulse to travel from the valve through the pipe line to tank and back at a speed of about 4000 ft. per sec. This effective portion for all but two of the water columns tested was the last 15% of the valve movement. For slower closures the pressures developed will be smaller than the maximum water hammer, the amount depending upon the resistance offered by the valve opening, the head of the line, etc.

10. For a slow closure the resistance through the valve opening is the chief force in stopping the mass of water.

11. In all but one of the water columns tested the resistance through the valve opening was not markedly larger than that at full opening until the valve had attained at least 85% of its closure. Since during the first 85% of the valve movement, little work is done in stopping the water, and since during this portion of the valve movement water hammer will not be developed even with rapid closing, it follows that it is immaterial how quickly this first 85% of closure is made. The time thus saved may well be used in lengthening the time for the remaining 15% of closure. The time-pressure diagram and the valve-movement-pressure diagram of one of the newer forms of water column illustrate the

hydraulic advantages which may be gained in this way. The effect in this case is the same as if a uniform closure five times as long as the actual time of closure were used. It is seen that making the last portion of the valve movement slowly is very advantageous.

12. With the time-pressure and valve-movement-pressure diagrams, and knowing the head, diameter, and length of the pipe line, the pressure which will be developed when the value is closed in a given time (slow closure) may be worked out approximately by the method given under the heading "17. Pressure Produced by Slow Closure of Valve."

13. When a relief valve is used as an auxiliary device, the size of the effective opening through the relief valve itself may be calculated from equation (18). Although relief valves offer a protection against excessive pressure, they do not of themselves fully solve the problem of stopping the mass of water. The type of water column having an internal relief valve which forms the seat for the cylindrical valve itself may be expected to give relief from high pressure with rapid closing, provided it is proportioned to furnish the proper resistance to flow to suit the conditions of flow head and length of pipe line.

14. From general considerations it is reasonable to expect that 8-in. and 12-in. water columns will have losses approximately the same as those found in the 10-in. water columns at the same velocity of flow. Losses based on this assumption, then, may be used until more accurate information is available without chance of serious error.

APPENDIX

20. *Diagrams for Friction Loss in New Cast-Iron Pipe.*—Although not a part of this investigation, the diagrams for friction loss in new cast-iron pipe prepared for the Committee on Water Service are reproduced here as a convenience to readers of the bulletin. Calculations for friction loss in pipes are at best somewhat uncertain, since the properties of flow are more or less indefinite. The roughness of the pipe, the workmanship in laying, and other variations in hydraulic conditions act to give rather wide variation of flow under the same head. This is shown in the results of carefully made experiments which are accepted as standard work. Moreover, the difficulty of making experiments on pipe of medium and large diameters at high velocities has prevented the accumulation of as complete a set of data as could be wished. The best that can be done is to use the formulas and tables which most closely conform to well established data. Fortunately, it is generally not necessary to know the friction loss within close limits.

In Fig. 11 and 12, the loss of head in feet per 100 ft. of new straight cast-iron pipe for the discharge indicated by the abscissas has been plotted to logarithmic scale. Fig. 11 is for pipe having internal diameters from 3 in. to 10 in. and Fig. 12 for pipe having internal diameters from 8 in. to 24. in., though the extreme sizes in each diagram have little range of discharge. With a little practice, interpolation between lines may be made fairly accurately.

The diagrams (Fig. 11 and 12) are based upon the formula for pipe friction, $h = \frac{fl}{d} \frac{v^2}{2g}$, where h is the head lost in friction in l ft. of pipe, d is the diameter in feet, and v is the velocity in feet per second; f is a factor depending upon the size and roughness of the pipe, and varying with the velocity of the water; g is the acceleration of gravity in feet per second per second. The values of the coefficient f used in preparing the diagrams are those given in Merriman's Treatise on Hydraulics. For the sizes above 6 in. the results are substantially the same as may be obtained from the formula: loss in 100 ft. $= 0.044 \frac{v^{1.8}}{d^{1.25}}$. The results for pipe of sizes 8 to 24 in. agree very closely with those obtained from the tables of Williams and Hazen for a pipe after approximately three years of service and also with the results of the excellent diagram given in Turneaure and Russell's Public Water Supplies. They also agree with the formula of Unwin, a noted English authority. The values found by the tables and diagrams just referred to do not differ by more than about 5 per cent from the values given by the diagram of Fig. 12 for the sizes named. It may be noted that this difference is much less than the variation in the results of experiments made at different times and places, and that a far greater difference may be expected with slight changes in the condition of roughness of the pipe. For sizes smaller than 8-in , the agreement of the several formulas is not so close. It is believed that the diagrams conform as closely to the results of experiments as any tables which have been published.

The values for head lost given by the diagrams for sizes from 8 to 24 inches diameter are generally smaller than those given by the Ellis and Howland tables and by the Darcy formula. A further difference is that these tables and formulas assume the head lost to vary as the square of the velocity, or nearly as the square, while the diagrams use 1.8 power. It seems that for high velocities, Ellis and Howland, and also Darcy, give a lost head which is higher than experiments show. The excellent form of the Ellis and Howland tables, as presented in the catalogs of the Addyston Pipe & Foundry Company, R. J. Wood & Co., and others, and the convenience of Weston's tables, which are based on the Darcy formula, have made these two sets of tables favorites for general use. It must be borne in mind, however, that for sizes above 6 inches, the lost head given by them for the higher velocities, is more than the actual loss.

21. *Diagrams for Velocity Head and Loss in Elbows and Tees.*— In Fig. 13 are plotted values of the velocity head $\frac{v^2}{2g}$ corresponding to discharges in gallons per minute through pipes having internal diameters of 8, 10, 12, 14, 16, and 18 inches. Although the scale is small, the values may be obtained with sufficient accuracy for the uses given in this bulletin.

With an assumed coefficient of entrance head, m, this diagram may also be used for finding the loss at entrance by merely multiplying the velocity head for the given discharge by the assumed coefficient. For ordinary forms of entrance this coefficient may be expected to be about 0.5, but it is not uncommon to use 1.0 for the coefficient, expecting that the larger value will cover other losses.

Fig. 13 also gives the loss of head in elbows of six sizes for given discharges in gallons per minute. The upper diagram is for elbows having a radius of curvature of axis of elbow equal to 1.5 diameters and the lower diagram for elbows having a radius equal to 3 diameters. The values are those selected by the Committee on Water Service as representing the maximum results of tests. Few data on large sizes and high velocities are available, but the values given in the diagrams are probably conservative and may be used until supplanted by more trustworthy information.

Fig. 13 also gives the loss of head in tees of six sizes for given discharges per minute, as presented by the Committee on Water Service. Information on the hydraulic properties of tees is even more incomplete than that of elbows, but the diagrams probably give the best results now available.

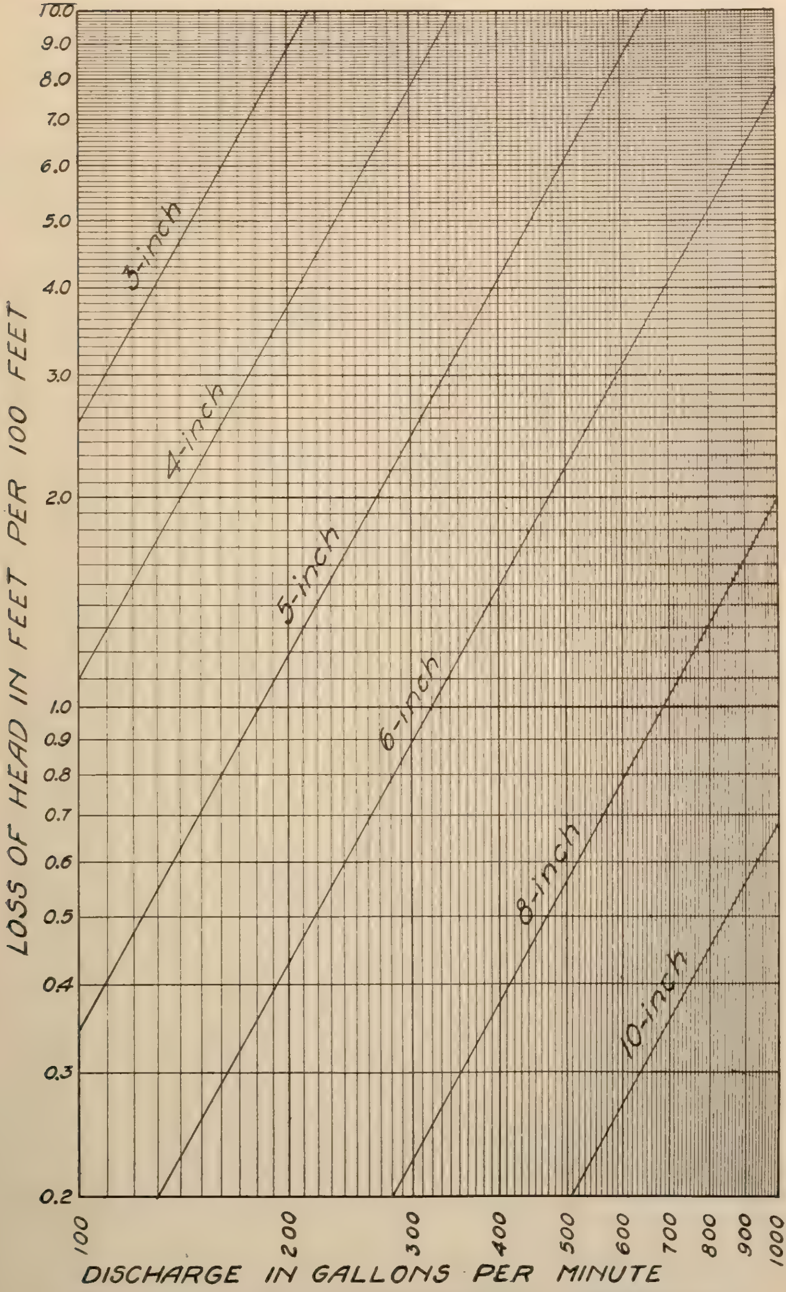

FIG. 11. LOSS OF HEAD IN NEW STRAIGHT CAST-IRON PIPE.

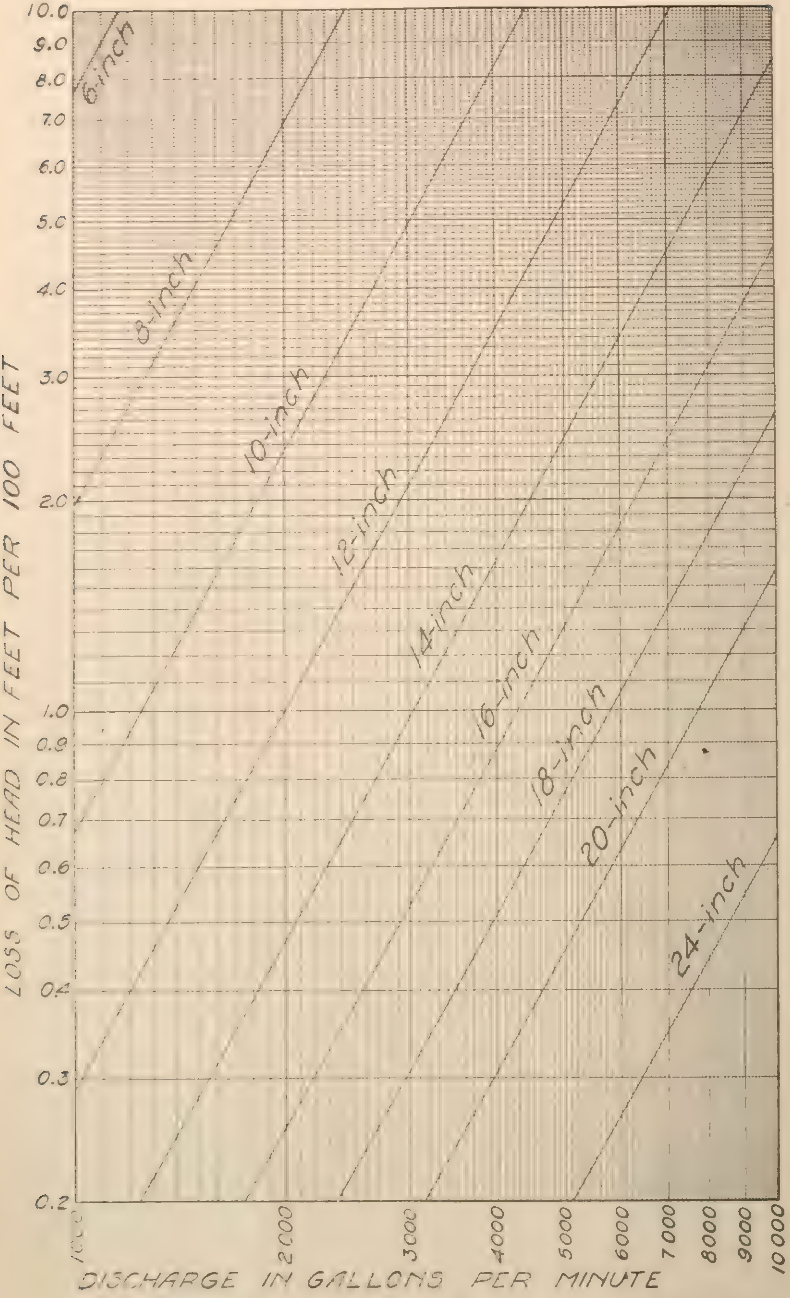

FIG. 12. LOSS OF HEAD IN NEW STRAIGHT CAST-IRON PIPE.

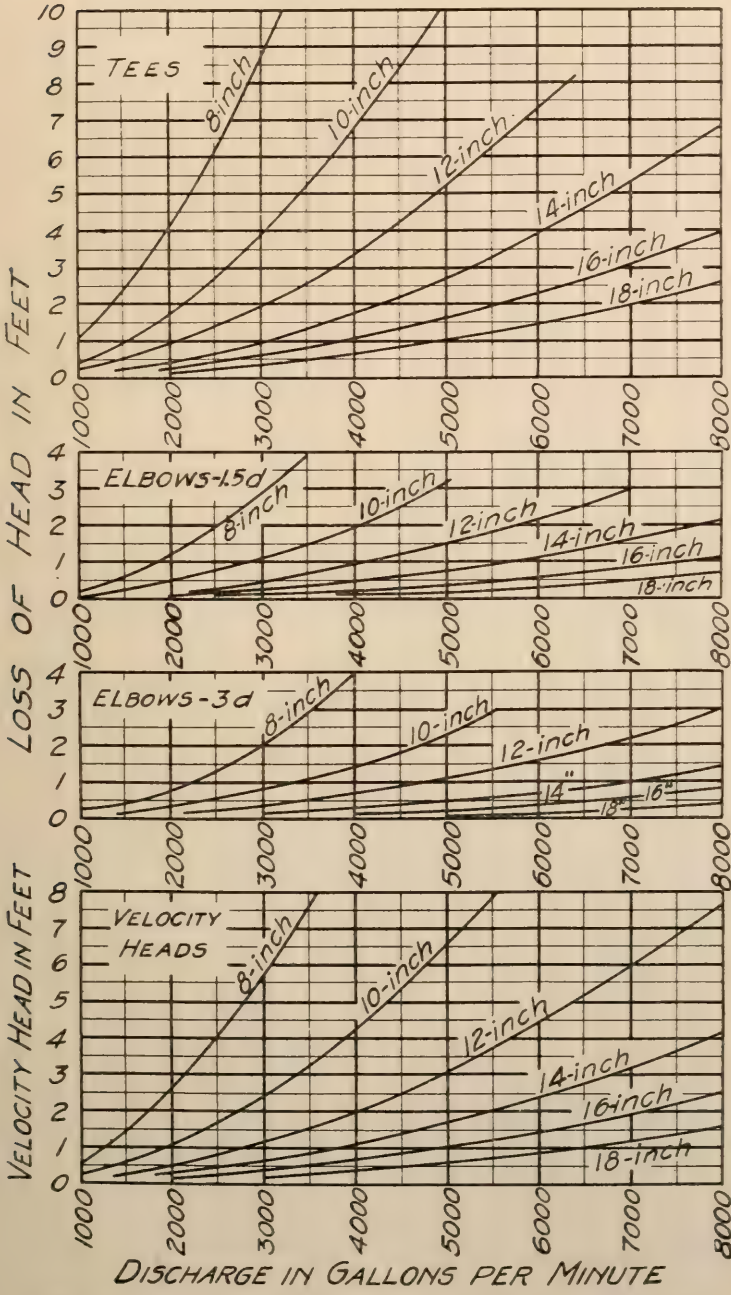

FIG. 13. DIAGRAMS FOR VELOCITY HEAD AND FOR LOSS OF HEAD IN ELBOWS AND TEES.

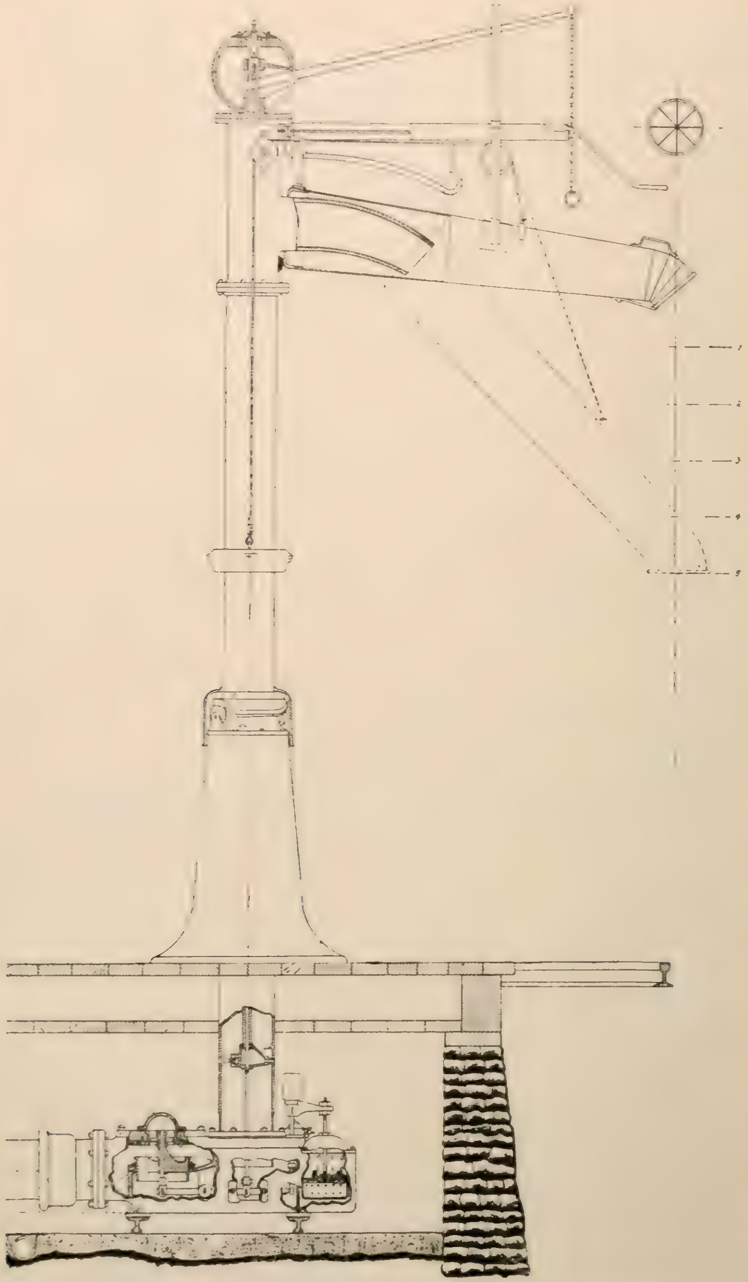

FIG. 14. WATER COLUMN I. (For details, see Fig. 42).

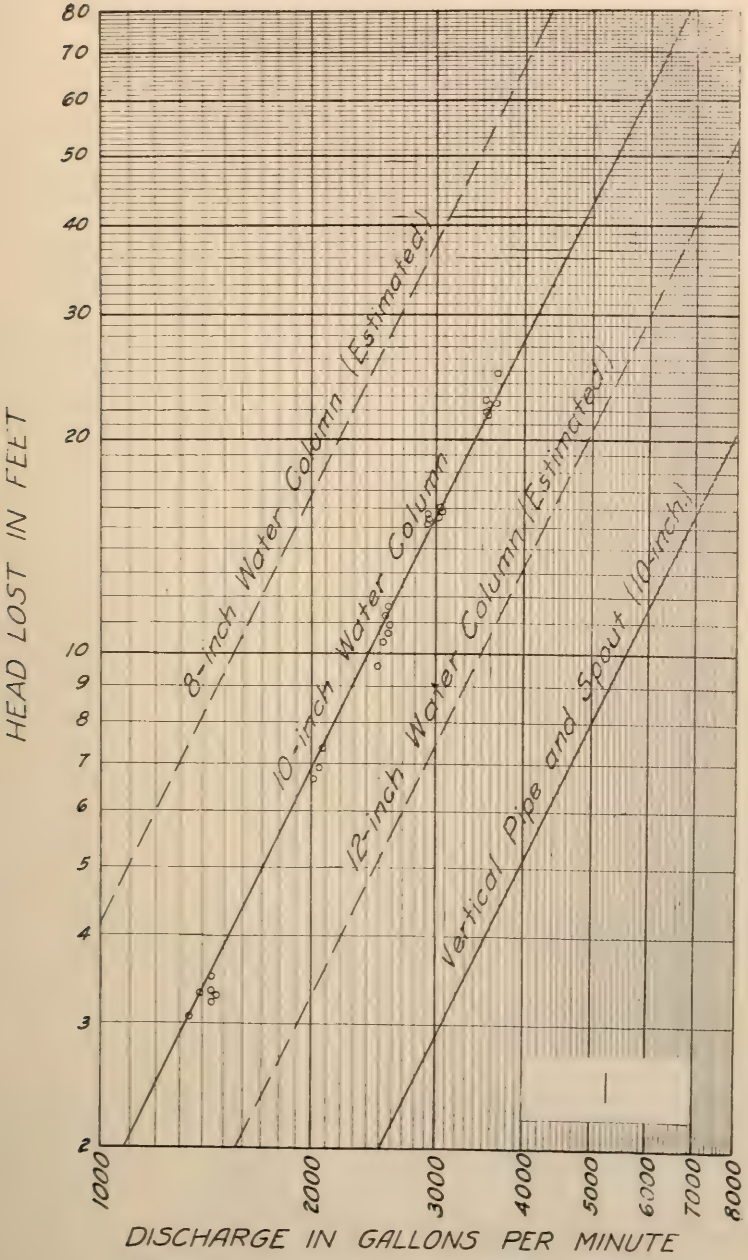

FIG. 15. LOSS OF HEAD IN WATER COLUMN I.

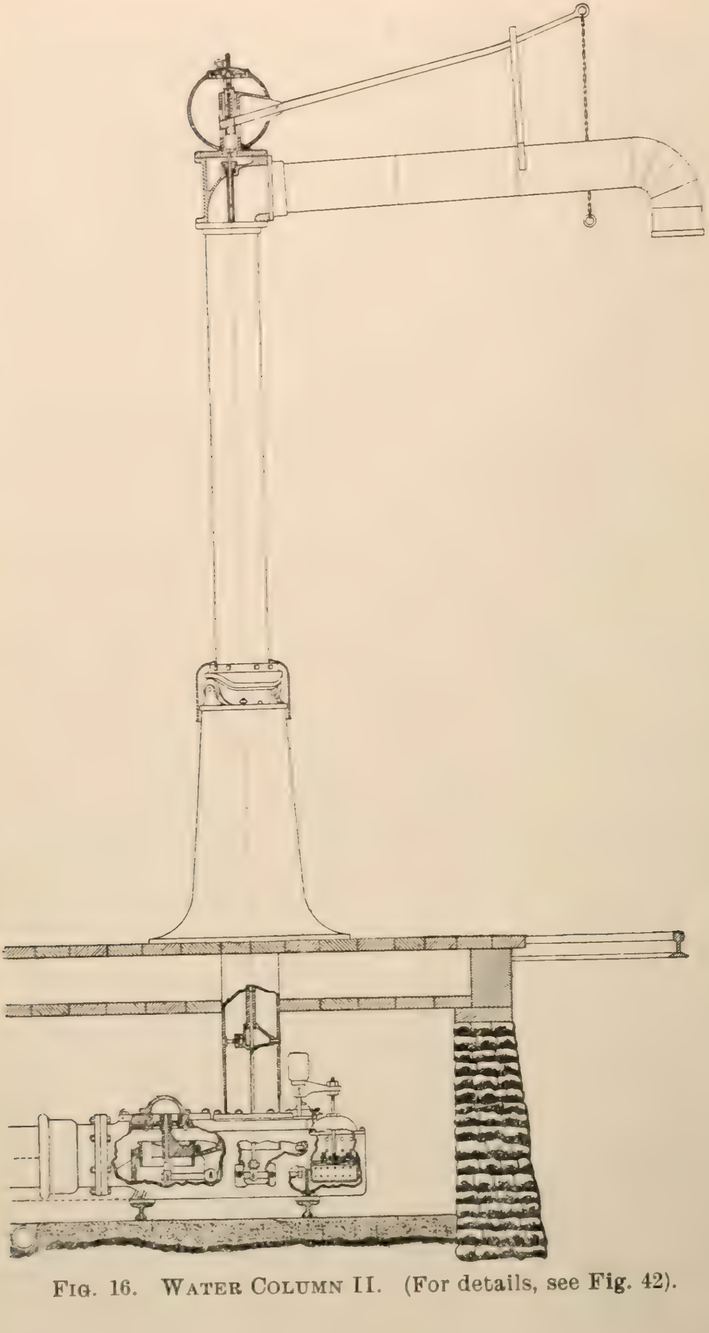

FIG. 16. WATER COLUMN II. (For details, see Fig. 42).

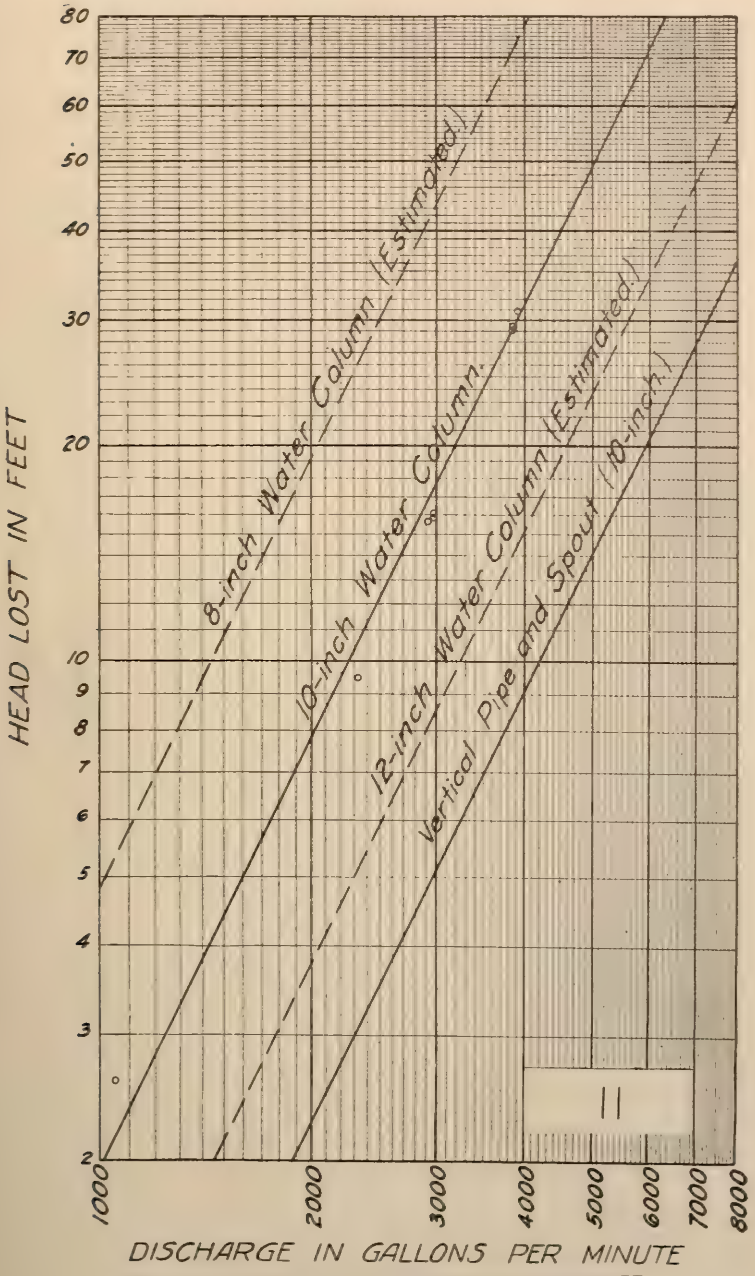

FIG. 17. LOSS OF HEAD IN WATER COLUMN II.

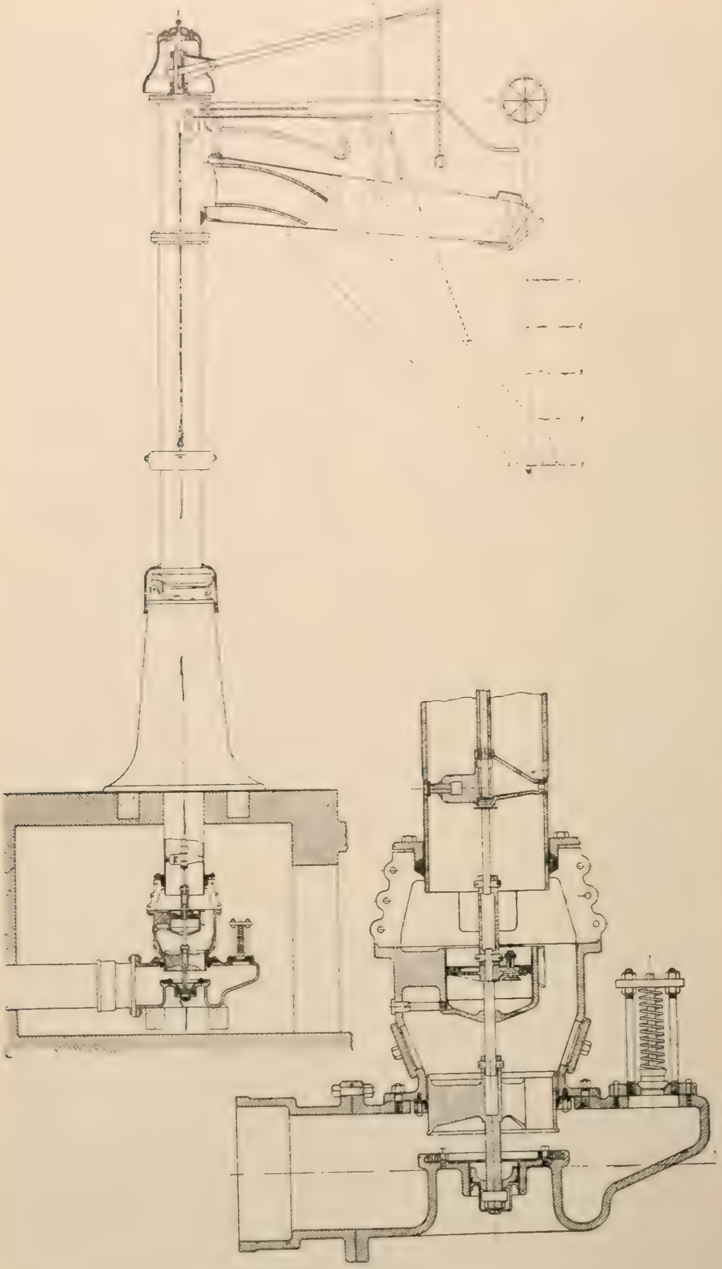

FIG. 18. WATER COLUMN III.

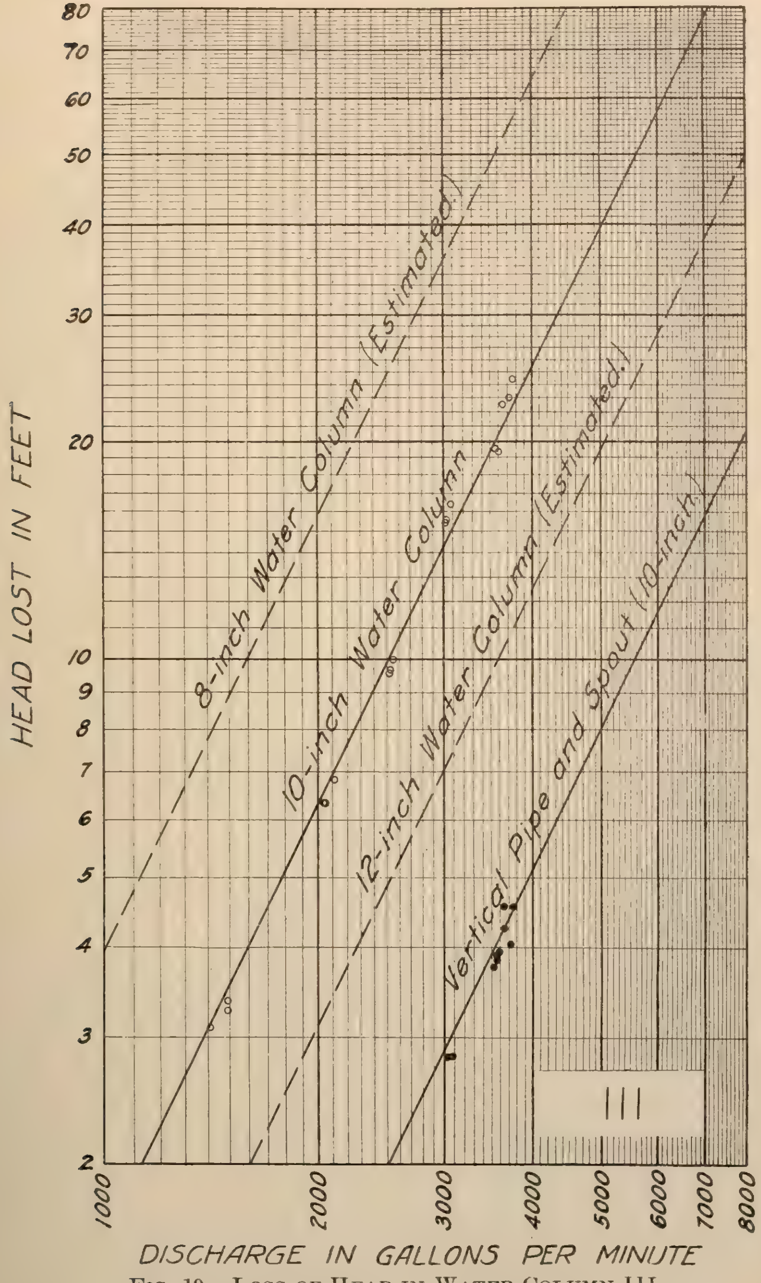

FIG. 19. LOSS OF HEAD IN WATER COLUMN III.

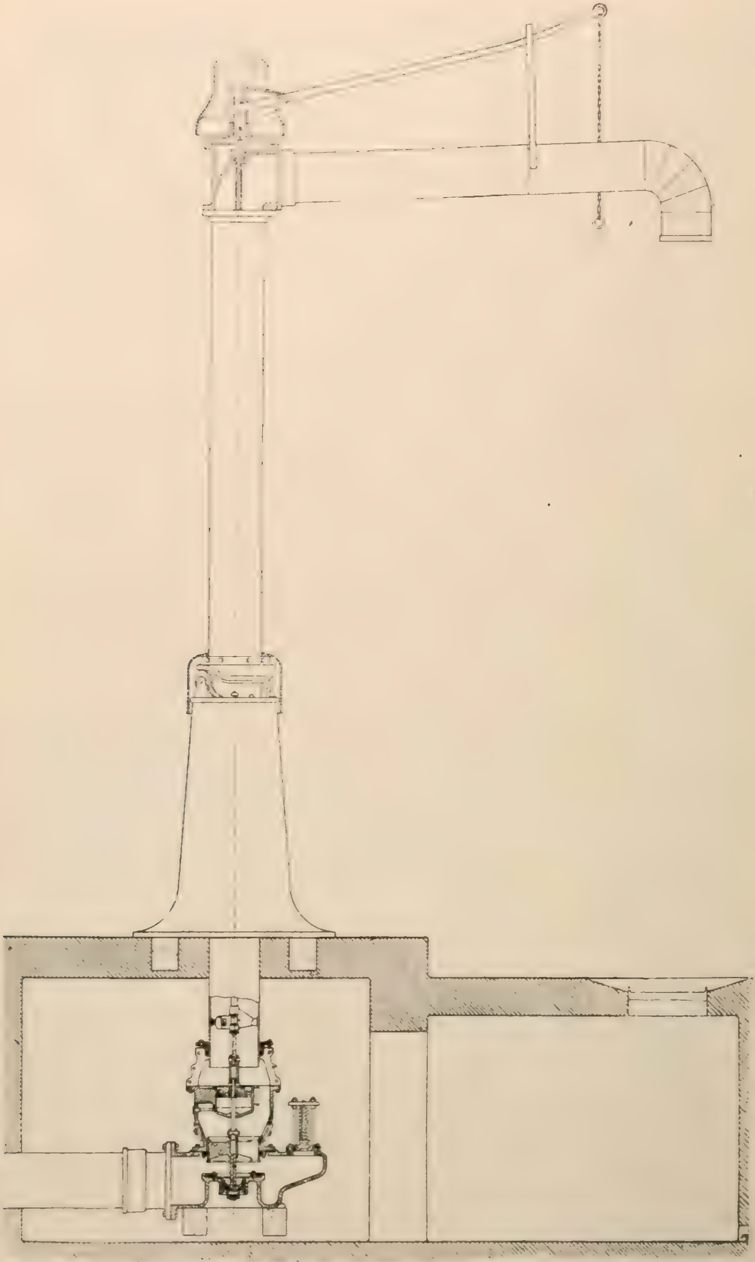

FIG. 20. WATER COLUMN IV.

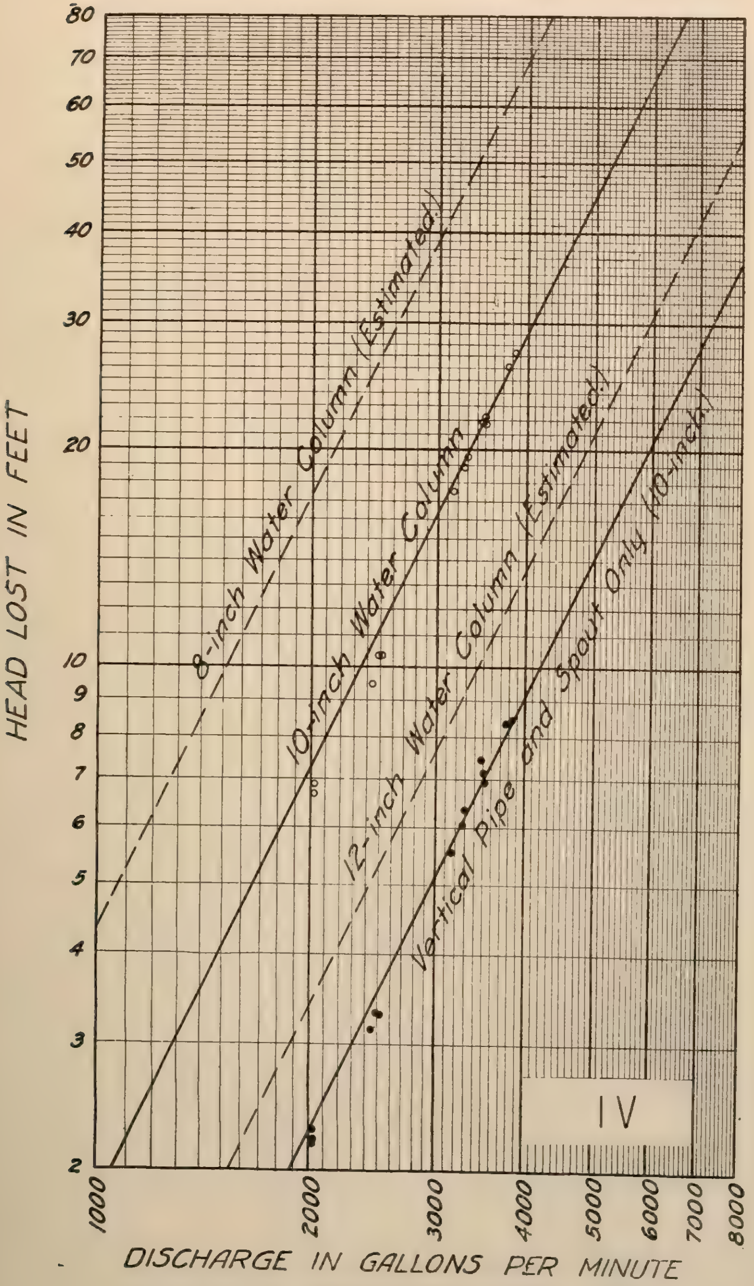

FIG. 21. LOSS OF HEAD IN WATER COLUMN IV.

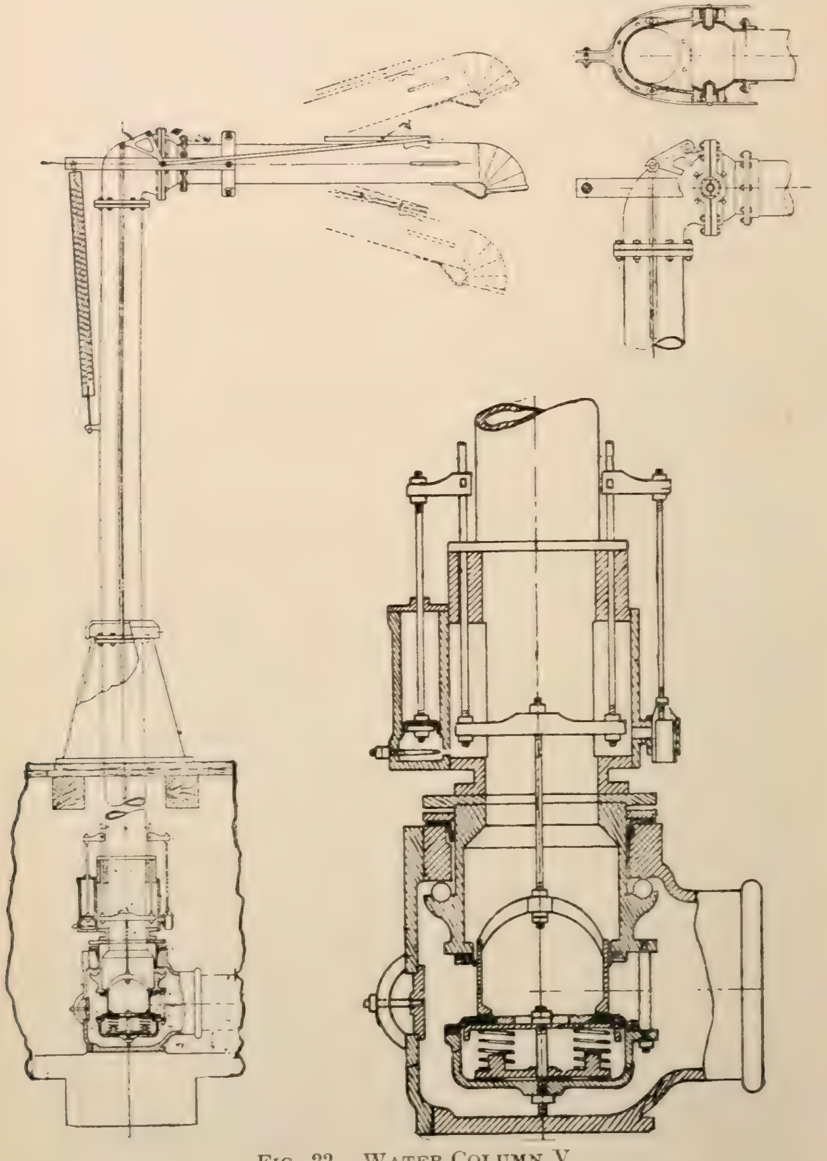

FIG. 22. WATER COLUMN V.

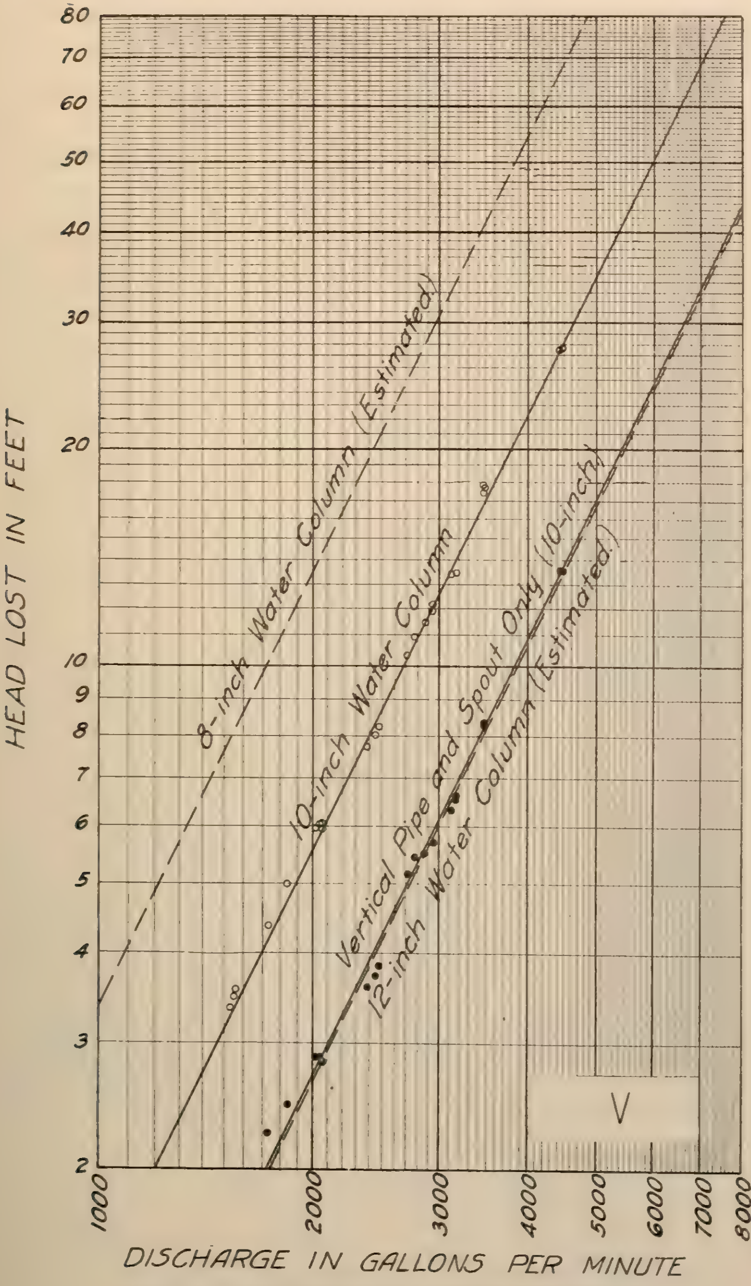

FIG. 23. LOSS OF HEAD IN WATER COLUMN V.

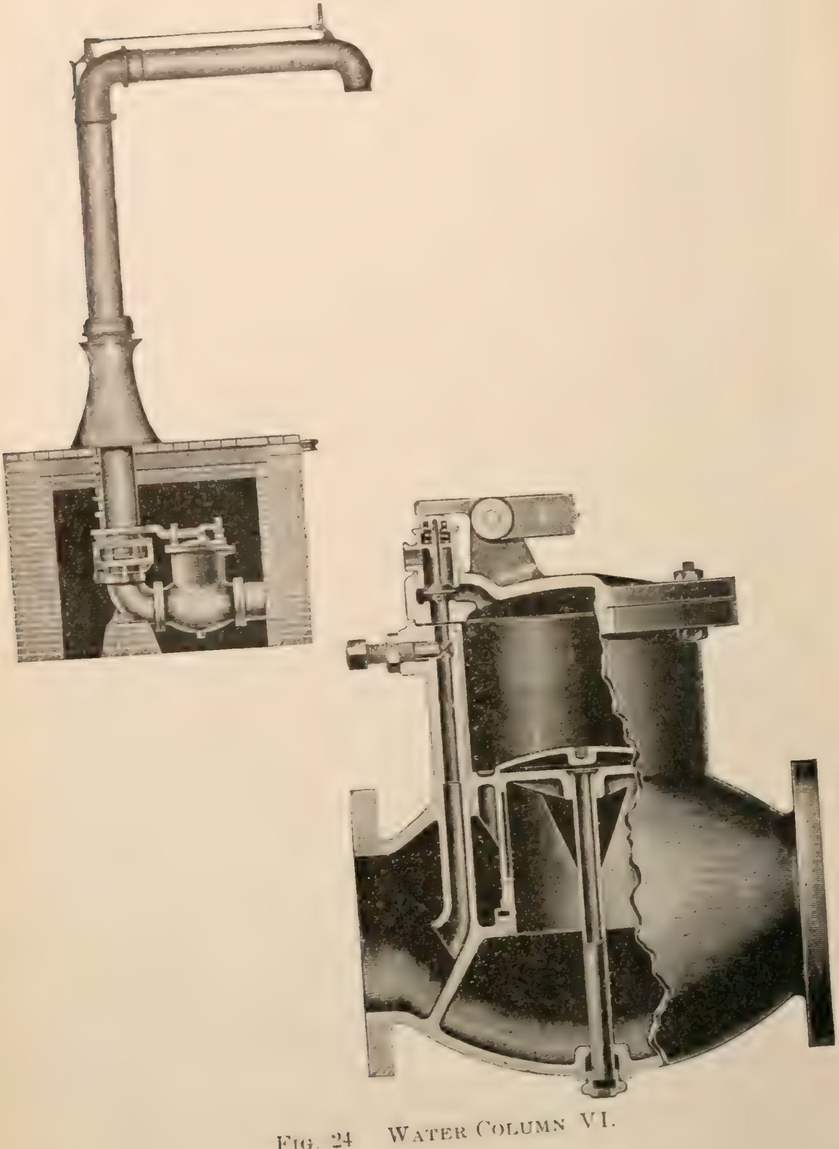

FIG. 24 WATER COLUMN VI.

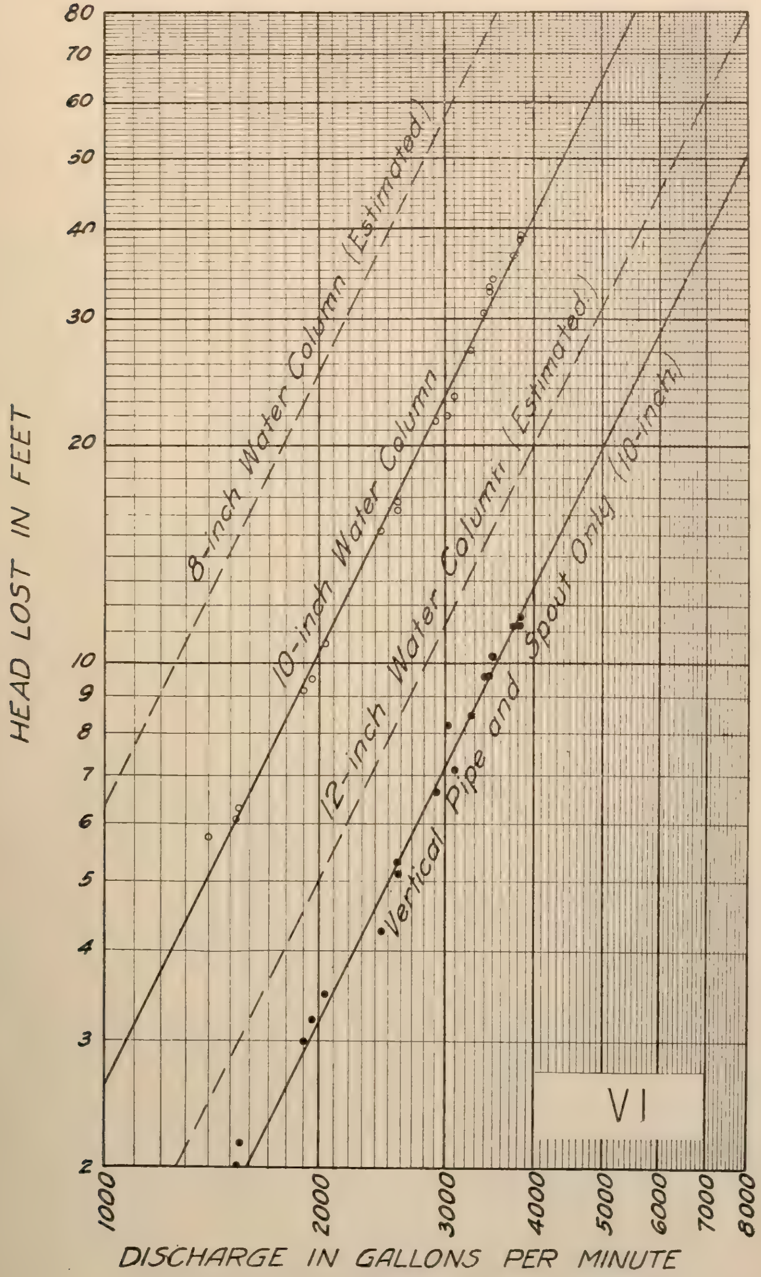

FIG. 25. LOSS OF HEAD IN WATER COLUMN VI.

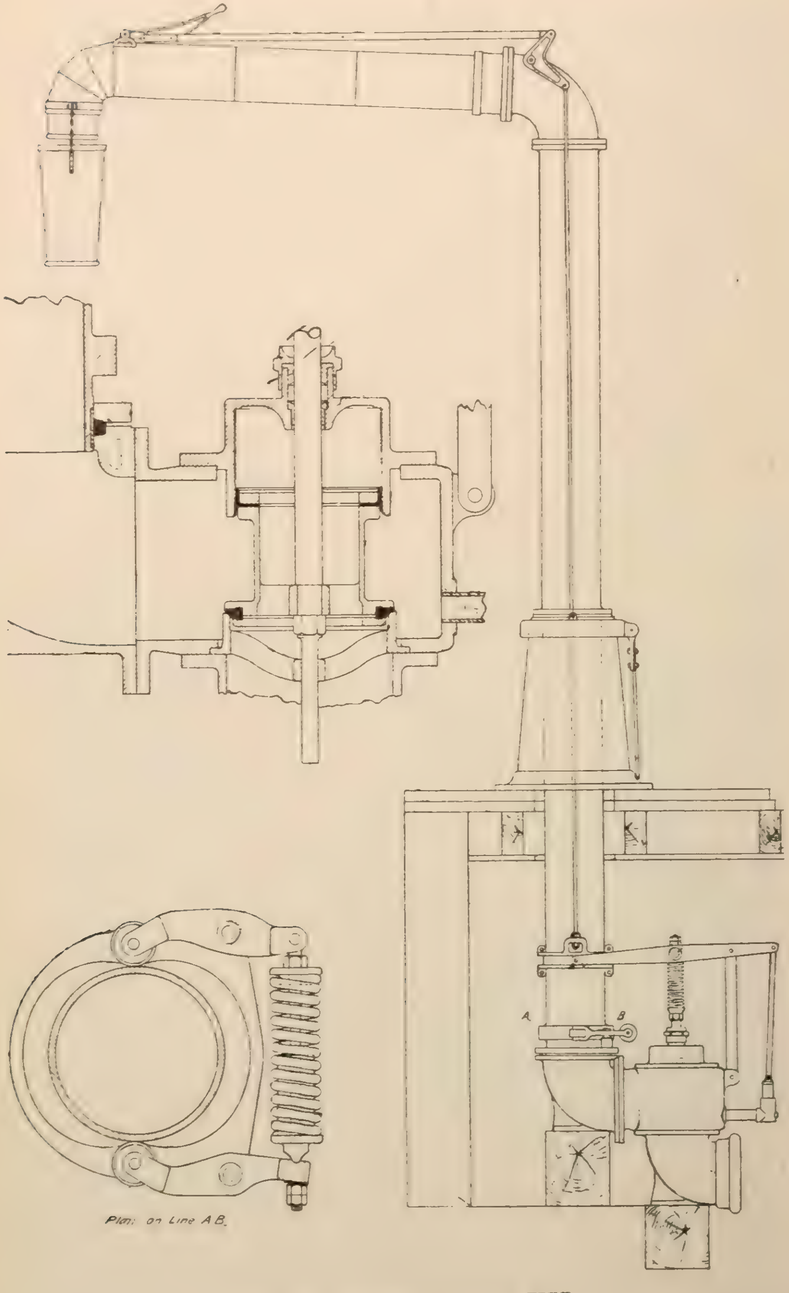

FIG. 26. WATER COLUMN VII.

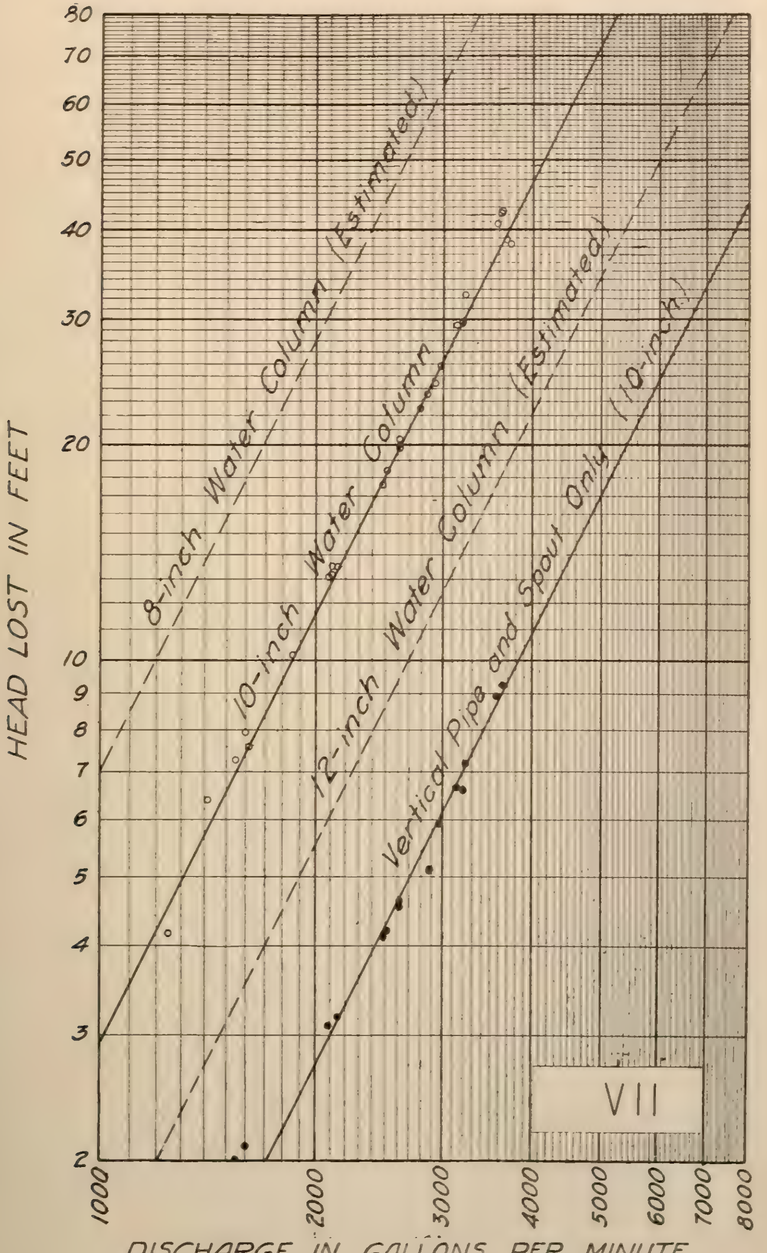

FIG. 27. LOSS OF HEAD IN WATER COLUMN VII.

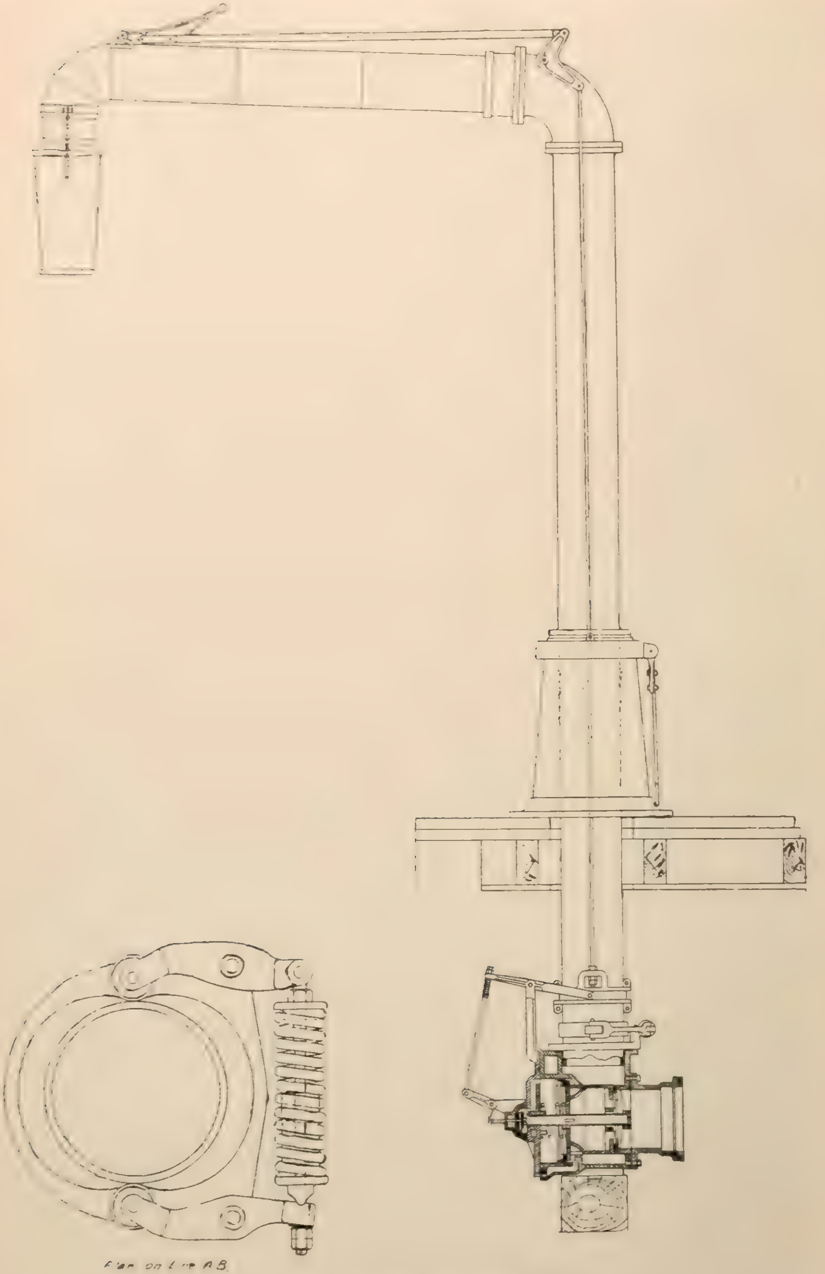

FIG. 28. WATER COLUMN VIII.

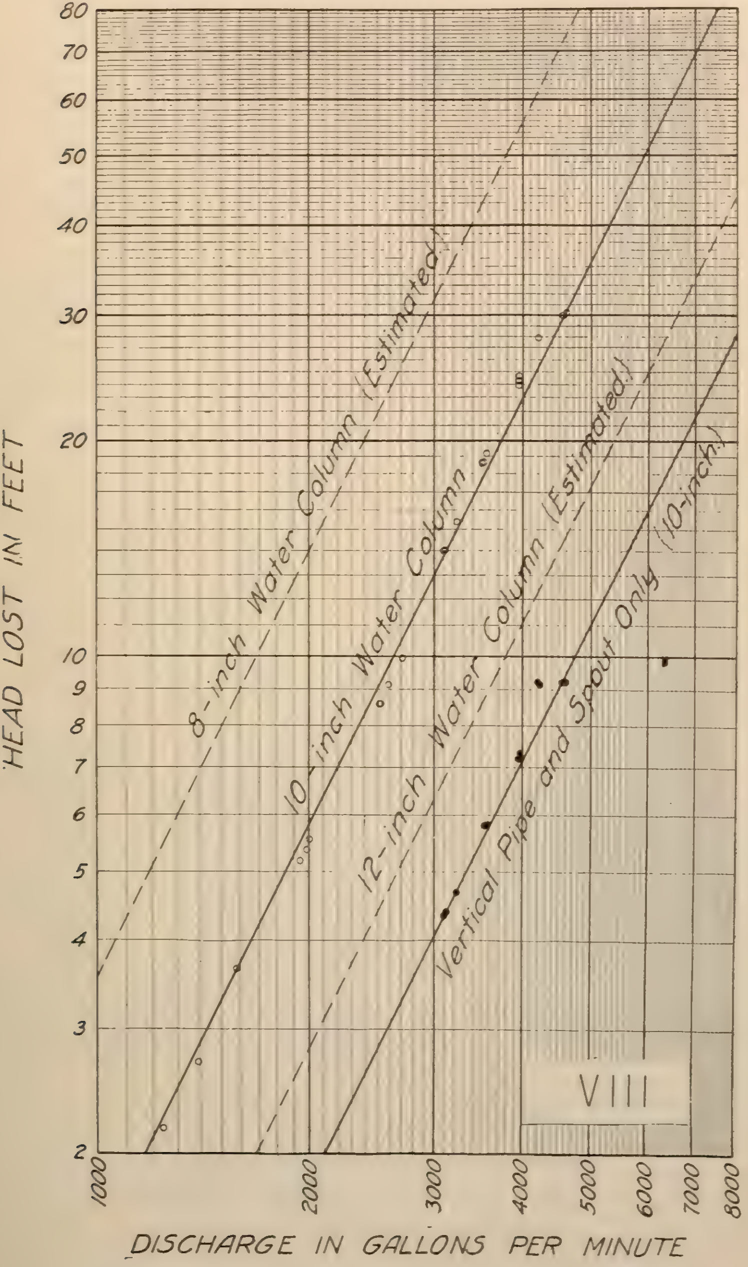

FIG. 29. LOSS OF HEAD IN WATER COLUMN VIII.

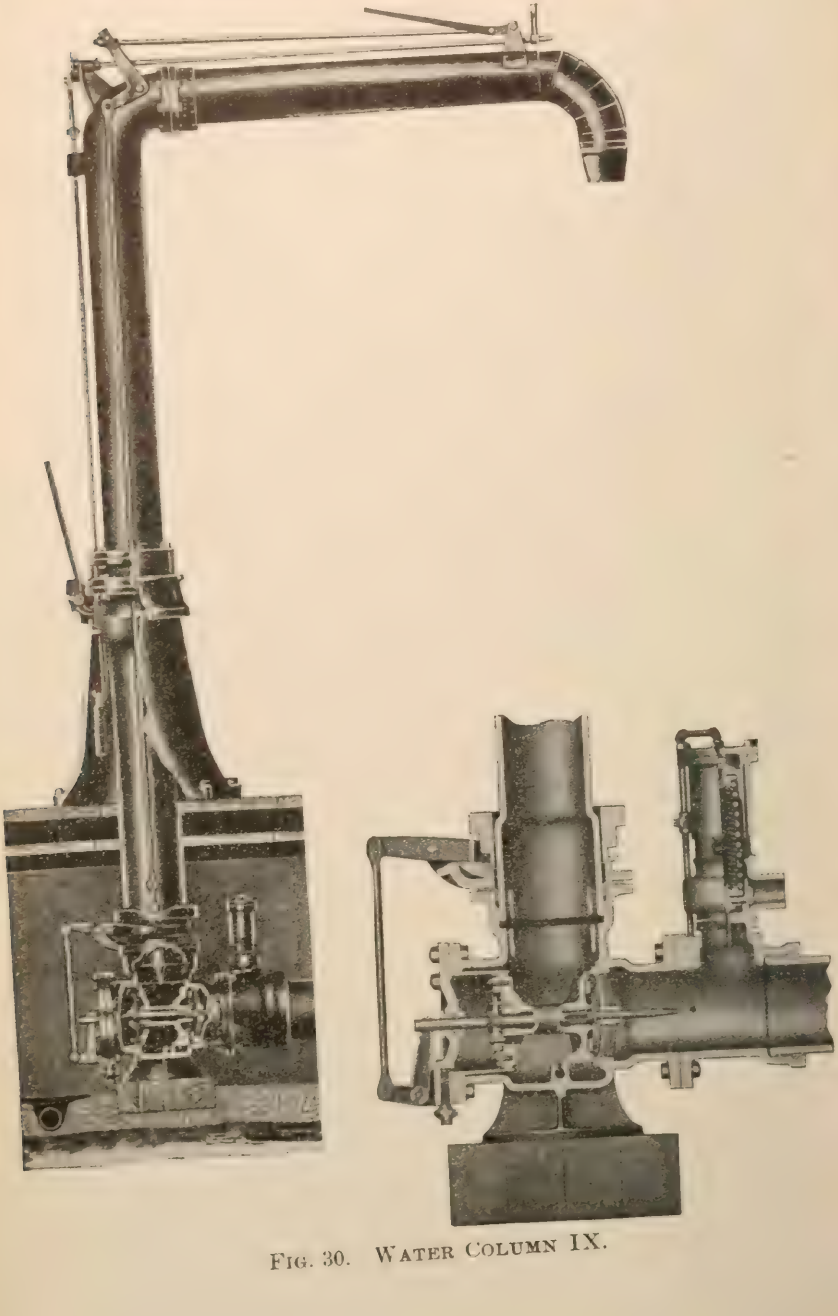

Fig. 30. Water Column IX.

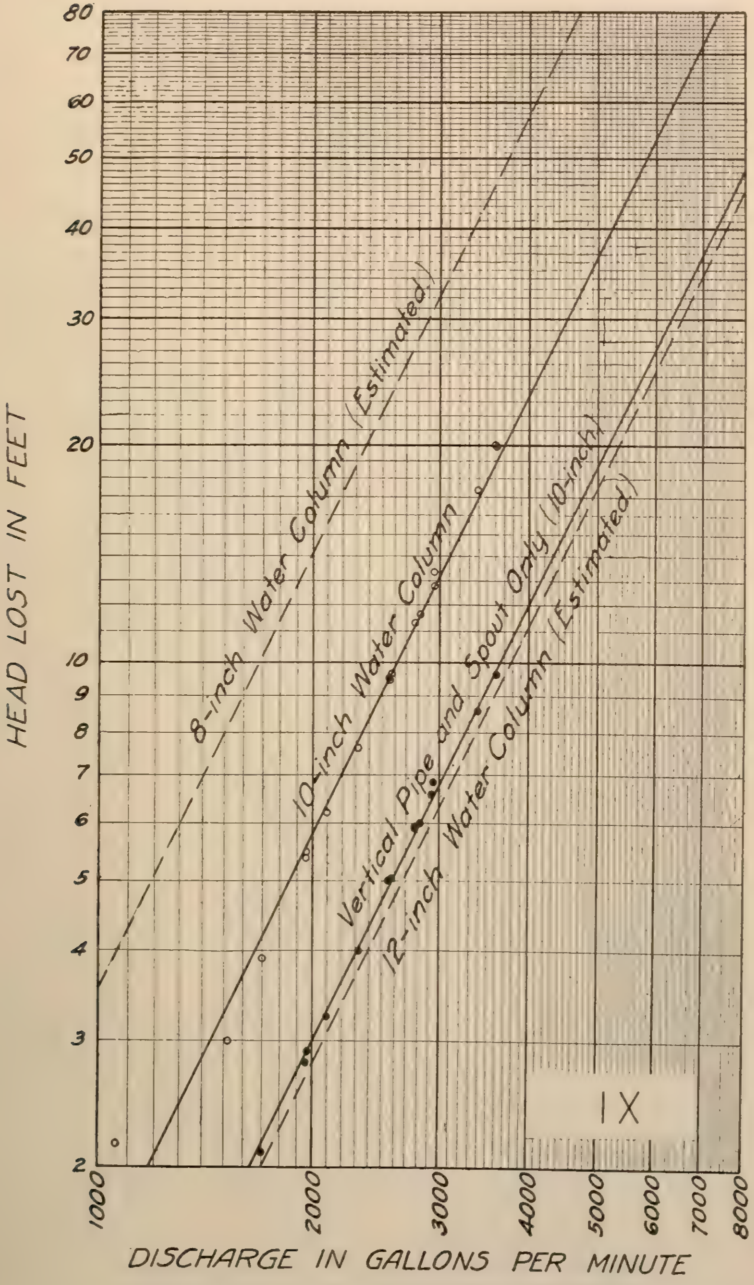

FIG. 31. LOSS OF HEAD IN WATER COLUMN IX.

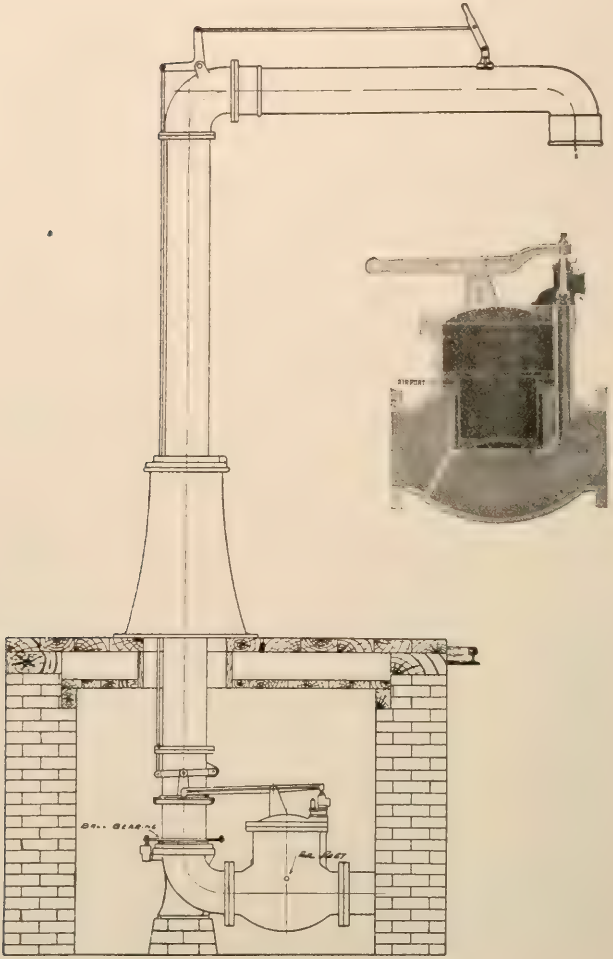

FIG. 32. WATER COLUMN X.

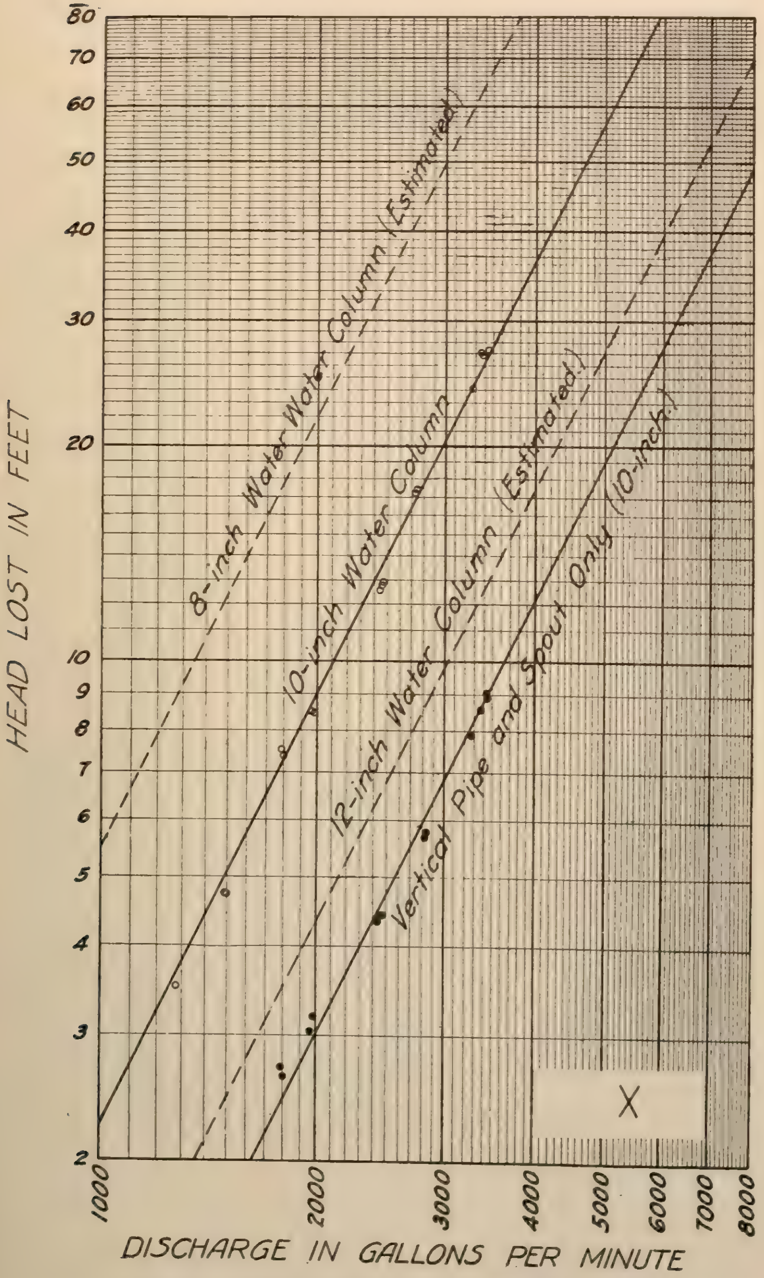

FIG. 33. LOSS OF HEAD IN WATER COLUMN X.

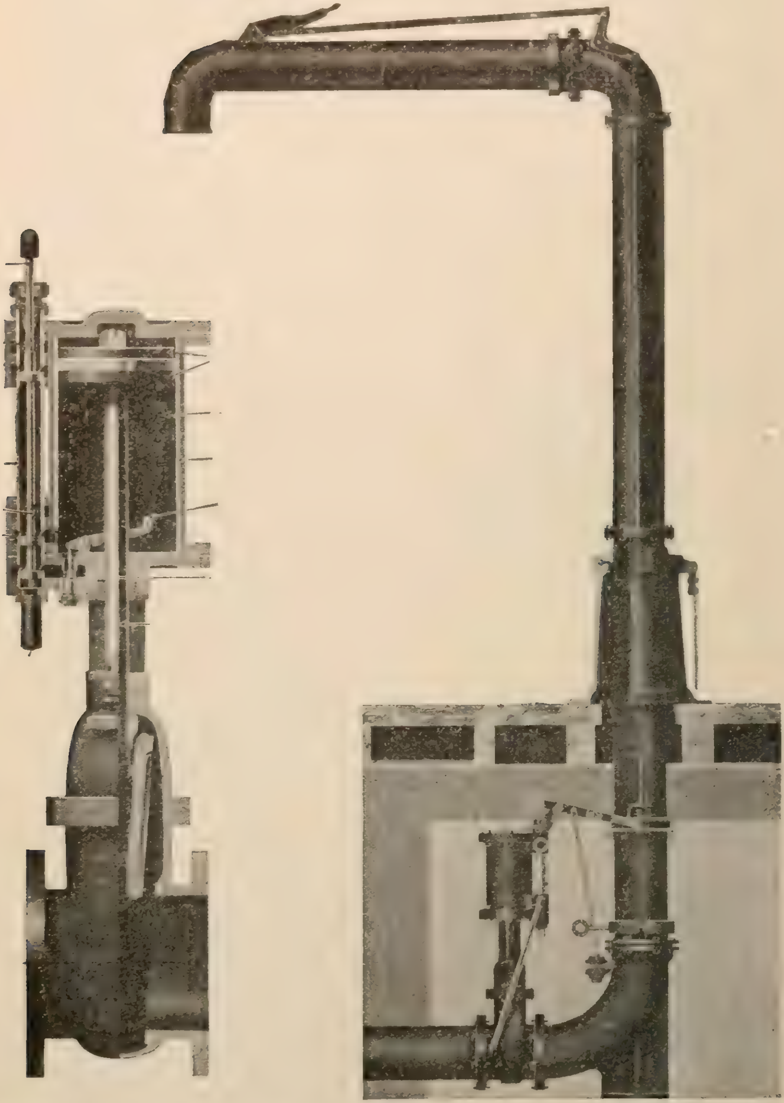

Fig. 34. Water Column XI. (In the test, the riser and spout of Water Column VI were used instead of those shown in the cut.)

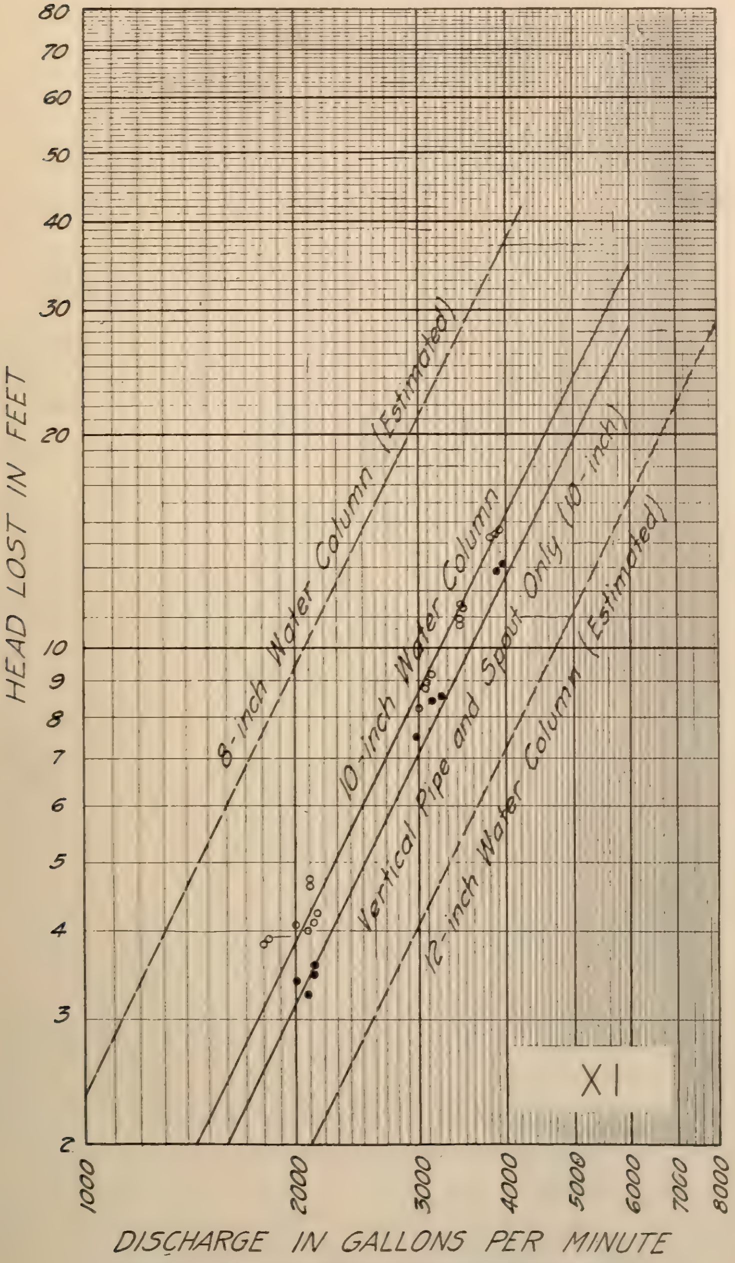

FIG. 35. LOSS OF HEAD IN WATER COLUMN XI.

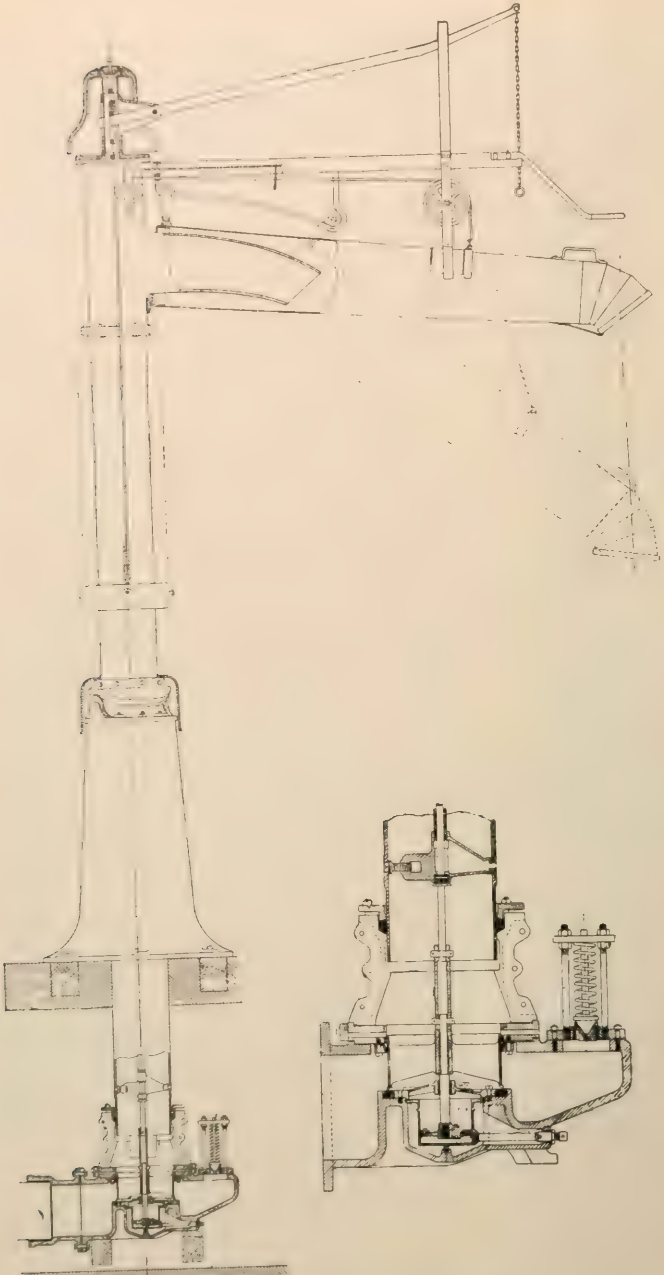

FIG. 36. WATER COLUMN XII.

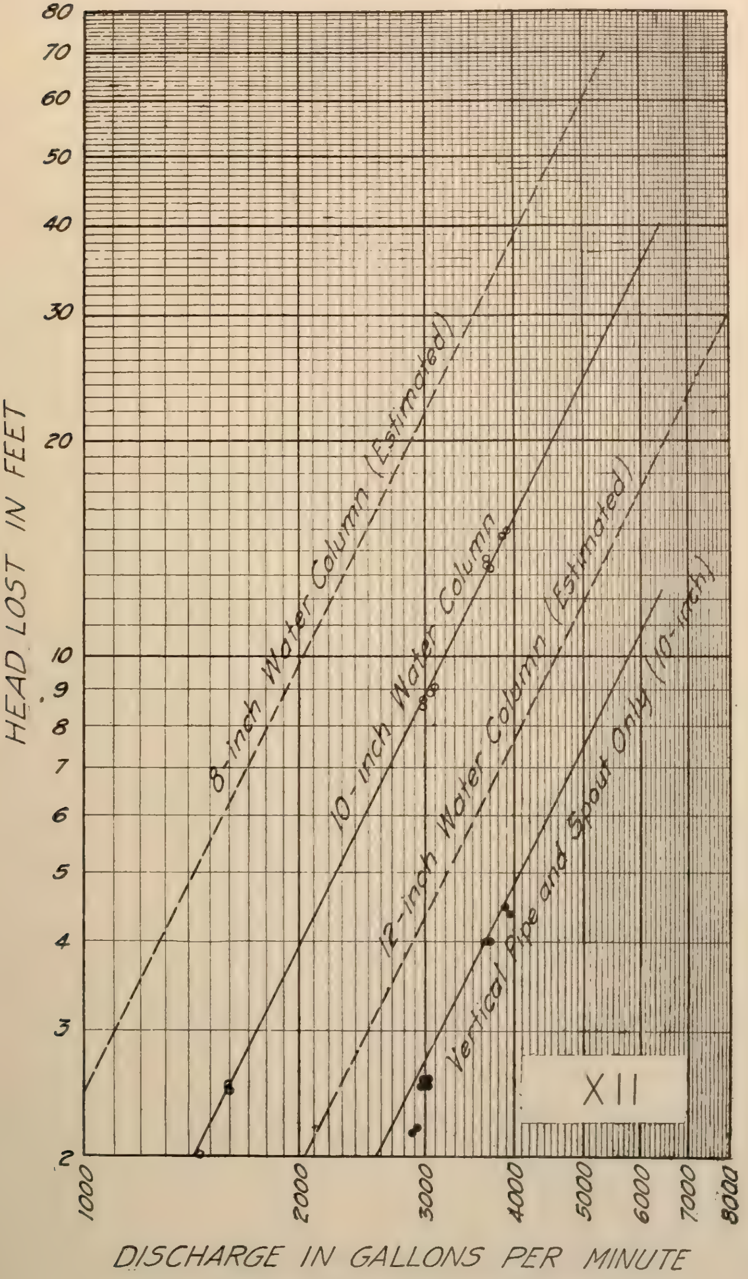

Fig. 37. Loss of Head in Water Column XII.

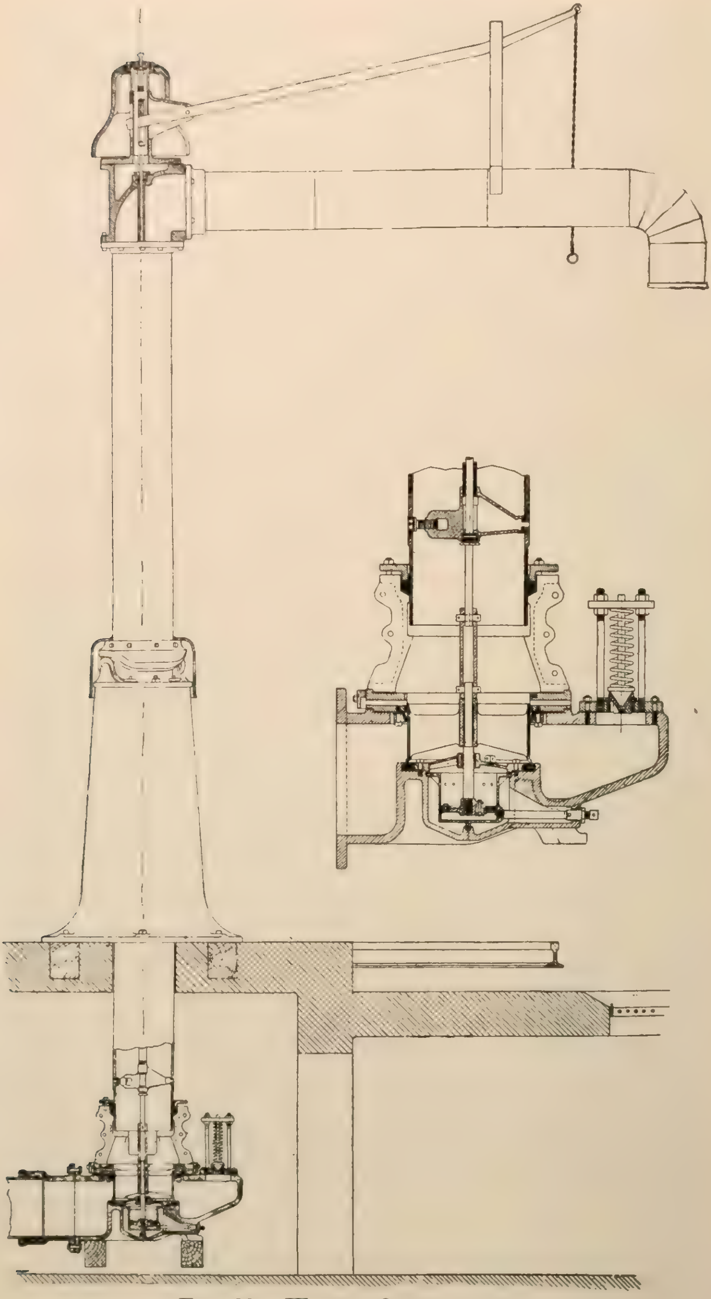

Fig. 38. Water Column XIII.

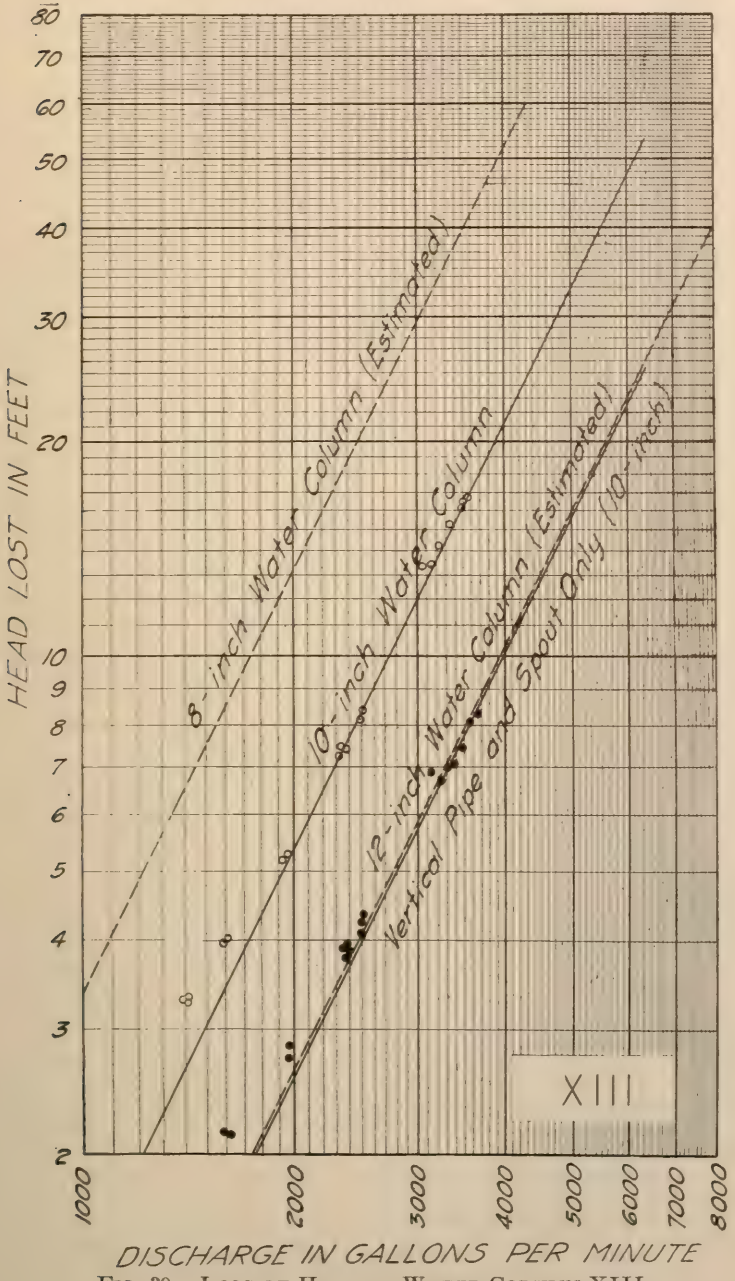

FIG. 39. LOSS OF HEAD IN WATER COLUMN XIII.

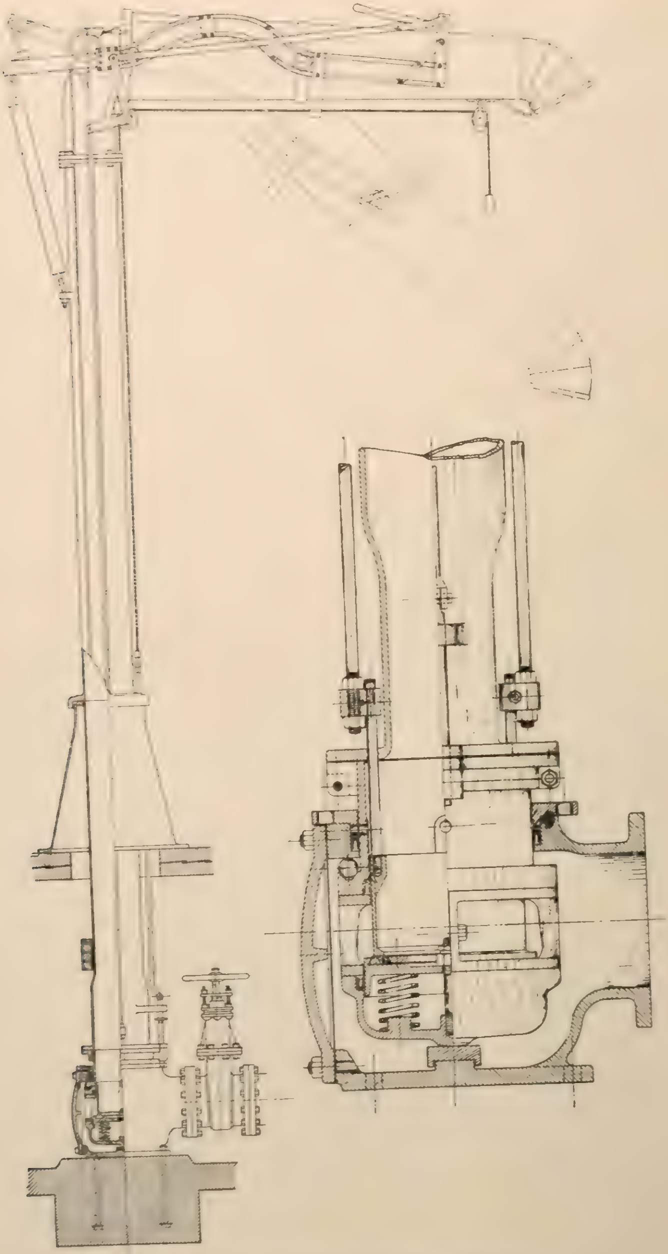

FIG. 40. WATER COLUMN XIV.

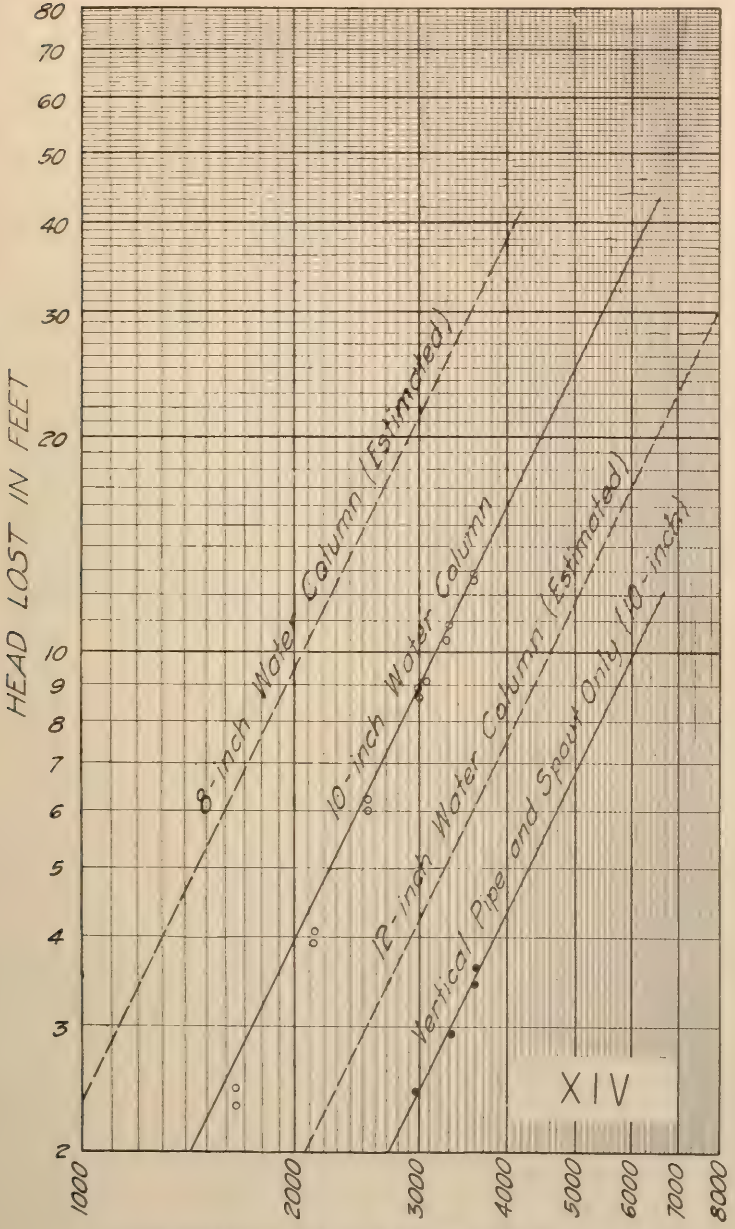

FIG. 41. LOSS OF HEAD IN WATER COLUMN XIV.

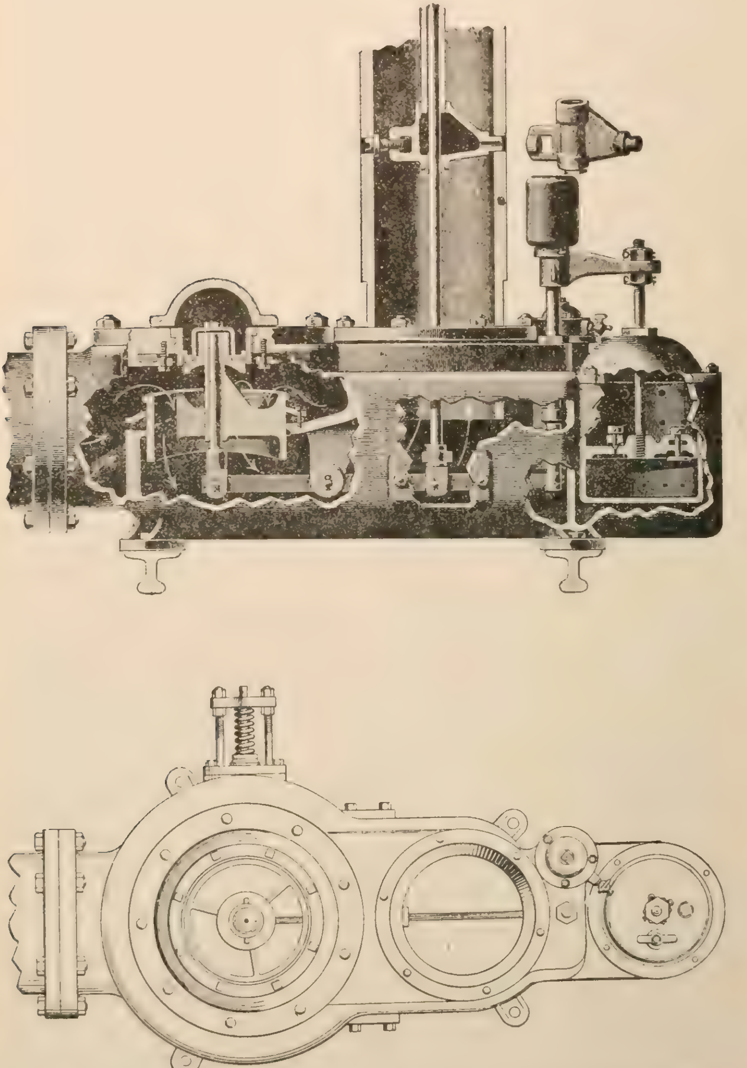

FIG. 42. DETAILS OF VALVE FOR WATER COLUMNS I AND II.

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UNIVERSITY OF ILLINOIS
ENGINEERING EXPERIMENT STATION

BULLETIN NO. 49 FEBRUARY 1911

TESTS OF NICKEL-STEEL RIVETED JOINTS

BY ARTHUR N. TALBOT, PROFESSOR OF MUNICIPAL AND SANITARY ENGINEERING AND IN CHARGE OF THEORETICAL AND APPLIED MECHANICS, AND HERBERT F. MOORE, ASSISTANT PROFESSOR IN THEORETICAL AND APPLIED MECHANICS

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TESTS OF NICKEL-STEEL RIVETED JOINTS.

I. Introduction.

1. *Preliminary.*—In designing riveted joints to connect structural members certain assumptions are made concerning the action of the rivets and other parts of the joint under load. Tests of riveted joints have not yet fully established the trustworthiness of some of these assumptions. In considering the design for a bridge across the St. Lawrence river near Quebec, Canada, to replace one which had failed during construction, the Board of Engineers of the Quebec Bridge appointed by the Canadian Government desired further experimental data on the action of riveted joints, especially as they had under consideration the use of nickel-steel plates and nickel-steel rivets; and arrangements were made by which tests were conducted at the University of Illinois. Later, the Pennsylvania Steel Company, of Steelton, Pennsylvania, arranged for similar tests of riveted joints made with chrome-nickel-steel plates and rivets. This bulletin records the results of the two series of tests. Although all material in the riveted joints tested contained nickel and the bulletin is termed Tests of Nickel-Steel Riveted Joints, in order to distinguish between the two sets the first set will be called nickel-steel riveted joints and the second set chrome-nickel-steel riveted joints.

The investigation included tests in tension and in alternate tension and compression. The bending of rivets, the deformation of the joints and the slip of the plates, the yield point of the joints, and the ultimate strength of the riveted joints were investigated. The tension tests included various forms of joint,–lap joints, butt joints, joints with fillers, joints with various numbers of rivets. The alternate tension and compression tests were upon one type of joint, a butt joint with very thick plates. A feature of interest was the repetitive loading of a joint alternately in tension and compression. The determination of the bending of the rivets was also a novel feature. The riveted joints which were tested in tension duplicated in dimensions a series of carbon-steel riveted joints tested for the American Railway Engineering and Maintenance of Way Association, and hence were not suited to develop the maximum strength possible with nickel-steel plates and rivets. To distinguish between the test pieces, the riveted joints of ordinary structural steel and ordinary rivets reported by the American Railway Engineering and Maintenance of Way Association will be called carbon-steel riveted joints; and those of the tests here recorded nickel-steel and chrome-nickel-steel riveted joints.

2. *The Strength of Riveted Joints.*—When a riveted joint is subjected to tension, tensile stresses are developed in the plates, bearing stresses in the rivets and in the plates at the rivet holes, shearing and bending stresses in the rivets, and shearing stresses in the plates beyond the rivet holes. Among the assumptions ordinarily made in the designing of riveted joints are: (a) that the tension and the tensile stresses are uniformly distributed throughout the width of the plate over the net section; (b) that the load is evenly divided among the rivets of the joint whether there are one or more rows and whether one row contains fewer rivets than another; (c) that little attention need be given to the relative amount of the stretch of the plate between the different rows of rivets; and (d) that the bearing area of a rivet is the product of its diameter (or the diameter of the hole) by the thickness of the plate in which it is placed. The limiting or safe shearing unit-stress in the rivet is ordinarily taken at two-thirds to three-fourths the limiting tensile unit-stress of the rivet material, and the bearing stress of the rivets, or of the plate, at one and one-third to one and one-half times this tensile stress. Under the assumed action of the joint, and considering the assumed limits of tensile, bearing, and shearing stresses, that stress which approaches most nearly to its assumed limit of stress at the load to be carried by the joint is considered to be the critical stress, whether this be the tensile stress in the net section of the plate, the bearing on the rivets, or the shearing stress in the various rivet sections. For any joint this critical stress will control the working strength of the joint; and whether it be tensile, bearing or shearing stress will depend upon the design of the joint and upon the ratios assumed between limiting or safe values of tensile stress, bearing stress and shearing stress.

For a given type of riveted joint the design in which the tensile stress, bearing stress, and shearing stress each reaches its limiting value, will have the highest efficiency, and a departure in proportions from this design will result in lowering the efficiency of the joint. However, matters connected with fabrication generally make modifications in design necessary—the size of rivet which may be handled with the facilities available, the size of hole which may properly be punched in the given thickness of metal, the closeness of spacing necessary to give tightness in joints subject to water or steam pressure, economies of workmanship, etc., will affect the design. Frequently empirical or conventional spacing or sizes of rivets are used. In any event the proper ratios of the allowable tensile, bearing, and shearing stresses may not be definitely known. The full action of the joint under load is uncertain. At the beginning of application of load the bearing may be on a small portion of the rivet surface. The rivet may not entirely fill the hole, and the shearing stress may not be uniformly distributed over the rivet section, or among all the rivets. Friction between plates by reason of the clamping action of the rivets may affect the distribution of stresses. Repetition of load may change the action of the joints and the amount of the stresses. Altogether, it is evident that fuller information on the action of riveted joints is necessary before the proper distribution of stresses may be judged. Conditions at rupture of the joint may not be applicable to working conditions, since stretch of plate, distortion of rivets, and the consequent redistribution of load beyond the yield point of the material may greatly change the action of the joint. It is evident that experimental data are needed to determine in detail the action of riveted joints under load.

3. *Existing Data.*—In the experimental study of the strength of riveted joints, the attention of English and American investigators has been directed largely to the resistance of joints to rupture; French, Dutch, and German investigators have given much attention to the frictional resistance to slip. Among European experiments those of Considere[1] in 1886, Dupuy[2], van der Kolk[3] in 1897, Bach[4] in 1894, and Pruess[5] in 1909, are worthy of special mention. The earlier of these investigators reported that the frictional hold of the rivets prevents slip in the joint up to loads slightly greater than those assumed in design practice for

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[1] Annales des Ponts et Chaussees, 1886.

[2] Annales des Ponts et Chaussees, 1895.

[3] Zeitschrift des Vereins Deutscher Ingenieure. June 26 and July 3, 1897. Le Genie Civil 1897. Practical Engineer, January 14, 1898, (Translation).

[4] Zeitschrift des Vereins Deutscher Ingenieure. 1892, p. 1142; 1894, p. 1231; 1895, p. 301.

[5] Iron and Steel Institute, Carnegie Scholarship Memoirs, Vol. 1 (1909), p. 60–95. In this memoir attention may be called to the fact that the shearing strength of specimens of rivet material (nickel steel) as given on page 76 should be divided by two, since the specimen is in double shear.

soft steel or wrought-iron plates and rivets. Considere, however, showed that if the frictional hold of the rivet is once broken the joint will afterwards slip under small loads if they are applied first in one direction and then in the other. In the experiments of Pruess (reported since the tests described in this bulletin were made) on joints having nickel-steel rivets, by the use of very sensitive extensometers, the existence of slip of joint was found at loads below those allowed in practice, and the test also showed that the use of nickel-steel rivets did not appreciably change the frictional hold of rivets on plates. All the European tests quoted above were made on joints in tension (with the exception of the few alternate stress tests of Considere) and, in general, were carried to rupture.

Among the English experiments may be mentioned those of Fairbairn[6], of Kirkaldy[7], and of Unwin and Kennedy[8]. In the experiments of Fairbairn and Kirkaldy the ultimate strength of the joint was studied. In the tests of Unwin and Kennedy a study was made of the effect of hand riveting and of hydraulic pressure riveting on both the ultimate strength and the frictional resistance of riveted joints.

In the United States the largest number of tests have been made at Watertown Arsenal[9]. In these tests deformation of joint was measured, but the efficiency of the joint was computed from the load carried at the ultimate.

In 1896 in the Laboratory of Applied Mechanics of the University of Illinois a series of tensile tests of riveted joints was made under the direction of A. N. Talbot by Van Ostrand and Thompson[10]. In these tests the slip of joint was found to begin at loads as low as those allowed in practice, and the effect of bending of rivet and of unequal distribution of stress in bearing of rivet and in shear was investigated.

In 1904 a series of tests on riveted joints in tension was carried on under the auspices of the Committee on Iron and Steel Structures of the American Railway Engineering and Maintenance of Way Association. The tests were made in the laboratory of the College of Civil Engineering of Cornell University[11]. In these tests the deformation of joint under load, the slip of the joint, and the permanent set of joint after load were measured, and the strength of joints with filler plates between main plates and cover plates was compared with that of joints without filler plates. These tests also showed the existence of slip at working loads.

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[6] Proceedings of the (British) Institution of Mechanical Engineers, April, 1881.

[7] American Machinist, May 11 and 18, 1893 (Condensed report of Kirkaldy's tests).

[8] Proceedings of the (British) Institution of Mechanical Engineers, 1881, 1882 and 1885.

[9] Tests of Metals (Watertown Arsenal Reports) 1885, 1886, 1887, 1896 and other volumes.

[10] Thesis of Van Ostrand and Thompson, 1896 (University of Illinois Library).

[11] Proceedings of the American Railway Engineering and Maintenance of Way Association, 1905, Vol. 6.

4. *Acknowledgment.*—The tests of nickel-steel riveted joints were undertaken at the request of the Board of Engineers of the Quebec Bridge, Messrs. H. E. Vautelet of Montreal, Maurice Fitzmaurice of London, England, and Ralph Modjeski of Chicago. The nickel-steel riveted joints were designed and furnished by the Board of Engineers and were manufactured under their inspection. Especial acknowledgment should be made to Mr. Ralph Modjeski for his suggestions and advice during the course of the work. The tests of chrome-nickel-steel riveted joints were undertaken at the request of the Pennsylvania Steel Company, the riveted joints following in form and dimensions the riveted joints tested for the Board of Engineers of the Quebec Bridge. Both the Board of Engineers of the Quebec Bridge and the Pennsylvania Steel Company agreed to the publication of the results of the tests by the Engineering Experiment Station of the University of Illinois. The expense of the tests was largely borne by the Board of Engineers and the Pennsylvania Steel Company.

The designing of the special auxiliary apparatus necessary for the tests and the work of testing was done by the research force of the Laboratory of Applied Mechanics of the University of Illinois. The chemical analyses of plate and rivet material were made in the laboratories of the Department of Applied Chemistry of the University of Illinois.

II. MATERIALS, TEST PIECES, AND METHOD OF TESTING.

5. *Materials.*—The plates of the joints included in this investigation were nickel steel and chrome nickel steel. In Table 1 are given the results of analyses of the steel. The carbon content and the manganese content were determined by colorimetric methods. Physical tests of several samples from different plates were made, and the results of these tests are given in Table 2.

The rivets were nickel steel and chrome nickel steel. The results of analyses of sample rivet material furnished by the Pennsylvania Steel Company as representative of the rivets used in making the joints are given in Table 1. Physical tests of this

TABLE 1.

CHEMICAL COMPOSITION OF RIVET AND PLATE MATERIAL.

Element	Nickel-steel Riveted Joints		Chrome-nickel-steel Riveted Joints	
	Rivet Material per cent	Plate Material per cent	Rivet Material per cent	Plate Material per cent
Carbon*	0.141	0.258	0.136	0.191
Sulphur	0.0023	0.008	0.038	0.035
Phosphorus	0.037	0.044	0.032	0.042
Manganese*	0.442	0.700	0.696	0.485
Nickel	3.33	3.330	0.986	0.733
Chromium	......	.....	0.240	0.170

* The carbon content and the manganese content were determined by colorimetric methods.

TABLE 2.

PHYSICAL PROPERTIES OF RIVET AND PLATE MATERIAL.

All stresses are given in pounds per square inch.

Item	Nickel-Steel		Chrome-Nickel-Steel	
	Rivet Material	Plate Material	Rivet Material	Plate Material
Number of specimens tested	2	9	2	8
Stress at first noticeable set*	......	38 200	.....	29 100
Stress at limit of proportionality of stress to deformation*	......	40 200	......	27 200
Stress at yield point +	45 000	51 700	38 400	36 300
Stress at ultimate	68 500	89 700	59 000	63 900
Elongation in 2 inches, per cent†	33.5	25.0	35.2	31.7
Reduction of area, per cent	63.4	55.8	63.3	59.9
Modulus of elasticity*	......	29 950 000	......	30 750 000

* Determined from tensile tests of threaded-end specimens with 8-inch gauge length.
+ Determined from tensile tests of threaded-end specimens with 2-inch gauge length.

rivet material were made, and the results of these tests are given in Table 2. A special series of tensile tests of flat specimens cut from plates of the chrome-nickel-steel riveted joints was made. Half of these specimens were tested at the Laboratory of Applied Mechanics of the University of Illinois and half at

Watertown Arsenal. Table 3 shows the principal results of these tests.

6. *Test Pieces.*—The form and dimensions of the riveted joints are shown in Fig. 1, 2, 3, 4, and 5. The joints tested in tension had the same form and dimensions as did the carbon-steel riveted joints in the series of tests made by the Maintenance of Way Association, which was noted on page 6, as the Board of Engineers desired, among other results, a comparison of the strength of carbon-steel and nickel-steel riveted joints. Fig. 5 shows the form and dimensions of the test joints used in the tests made under alternate tension and compression. The plates were made thick to avoid buckling under compression.

TABLE 3.

TENSILE TESTS OF FLAT SPECIMENS OF CHROME-NICKEL STEEL.

Item	University of Illinois Tests	Watertown Arsenal Tests
Number of specimens tested	6	6
Stress at yield point pounds per square inch..	36 500	36 800
Stress at ultimate pounds per square inch..	60 000	61 000
Elongation, per cent........	27.3*	27.1†
Reduction of area, per cent..	54.0	51.9

* Gauge length eight inches.
† Gauge length six inches.

The joints were fabricated by the Pennsylvania Steel Company at their works at Steelton, Pennsylvania. An effort was made to have the quality of the riveting representative of high-grade commercial work—made neither too well nor too poorly. The nickel-steel test pieces were made under the inspection of one of Mr. Modjeski's inspectors, acting for the Board of Engineers of the Quebec Bridge; and the inspector rejected some joints as being more tightly riveted than could be expected under ordinary conditions of structural practice. The Pennsylvania Steel Company gave their own supervision in the making of the chrome-nickel-steel test pieces.

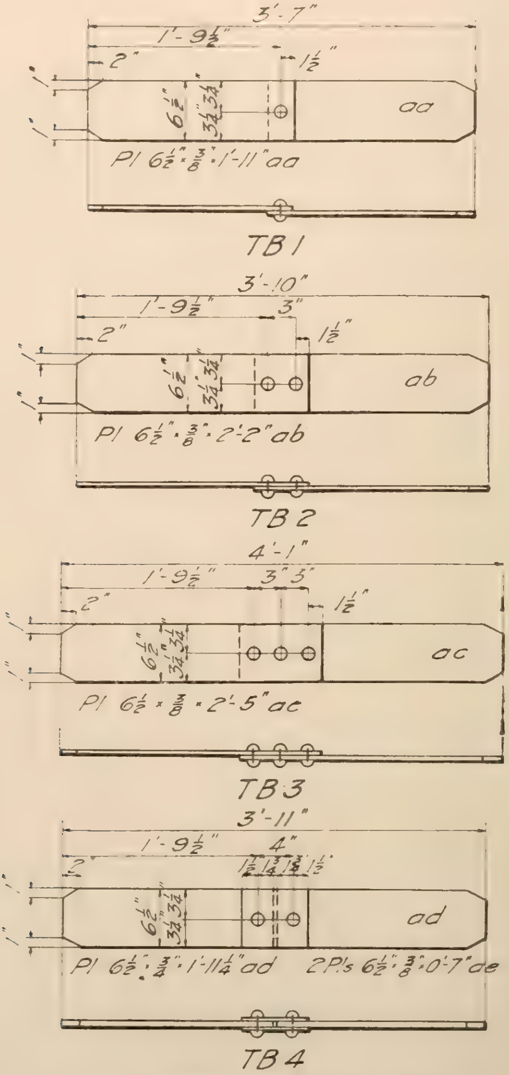
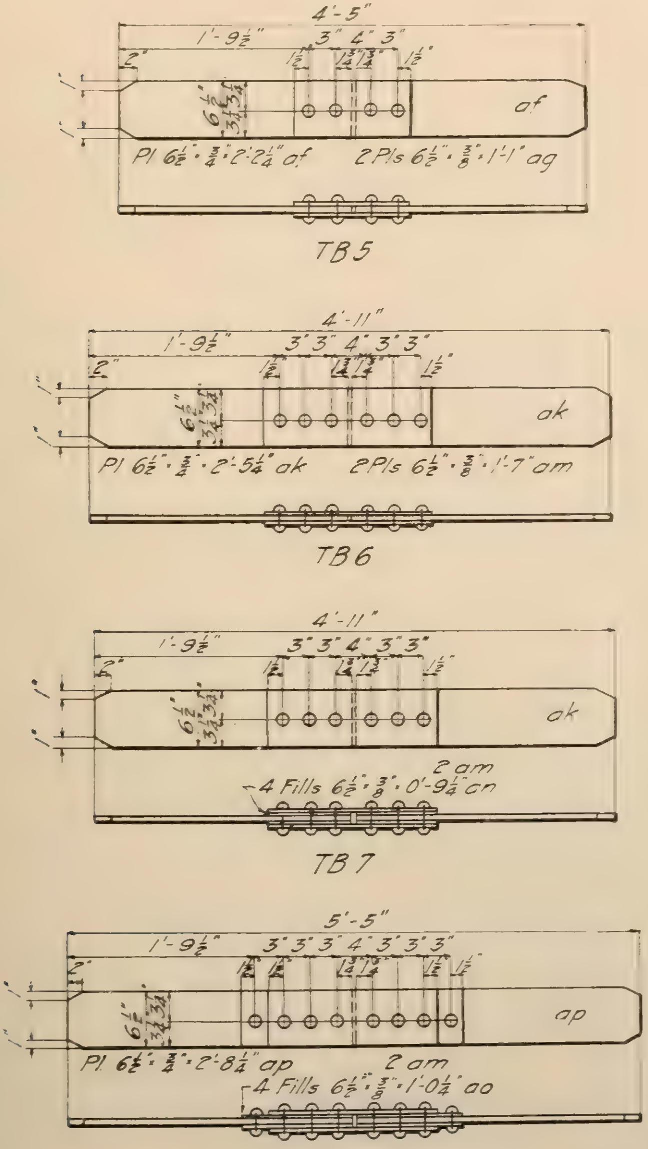


Fig. 1. Riveted Joint Test Pieces.



TB 5

TB 6

TB 7

TB 8

FIG. 2. RIVETED JOINT TEST PIECES.

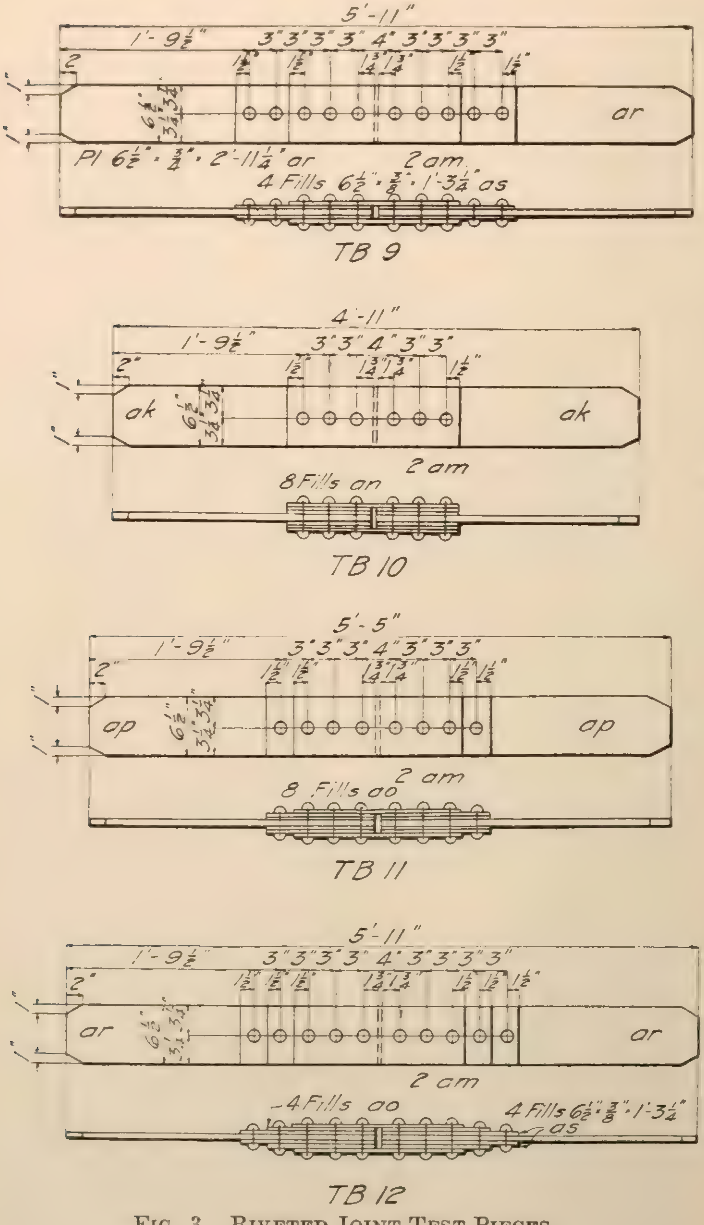


FIG. 3. RIVETED JOINT TEST PIECES.

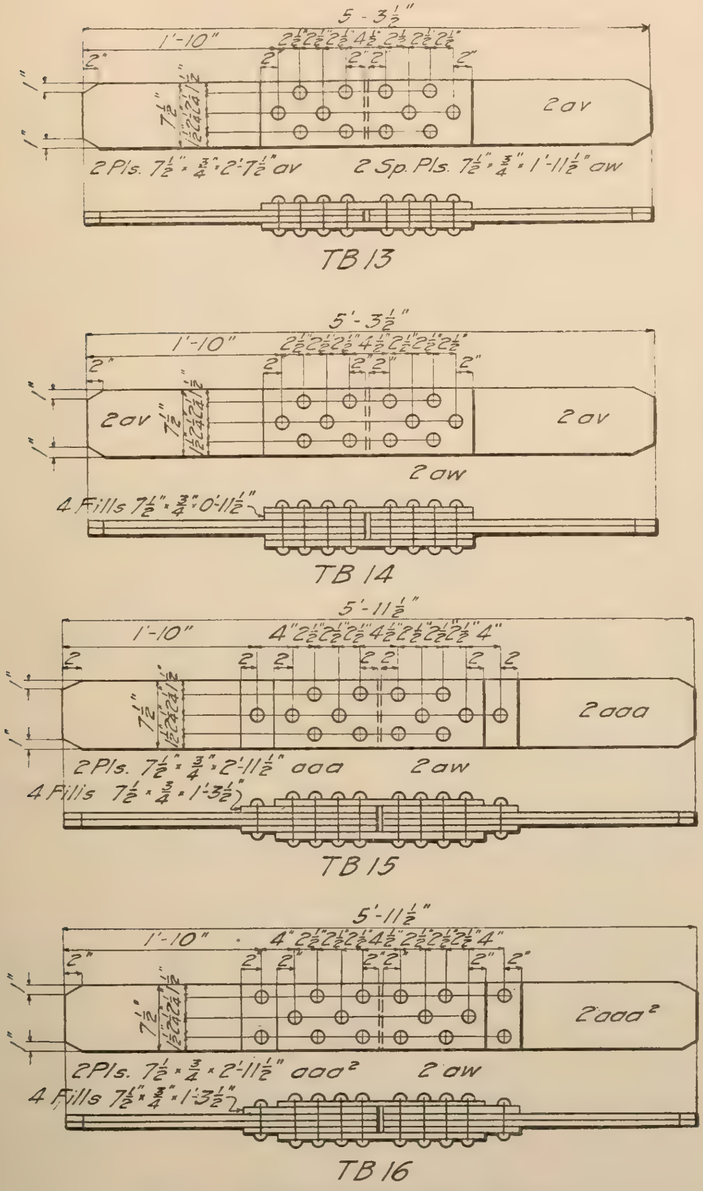
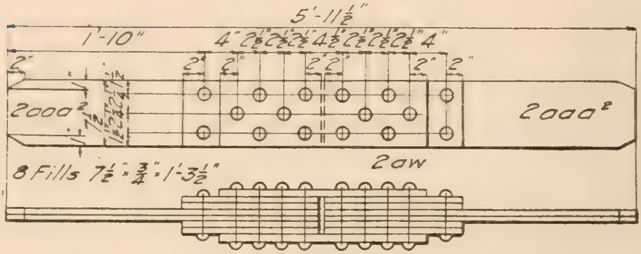


FIG. 4. RIVETED JOINT TEST PIECES.



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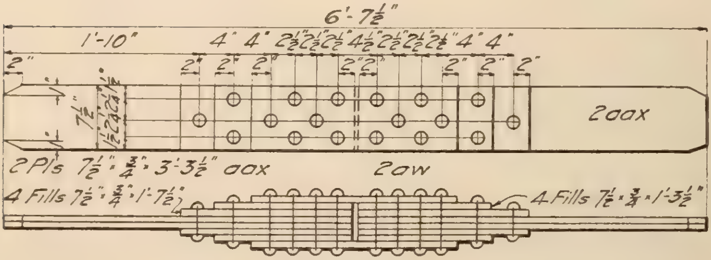

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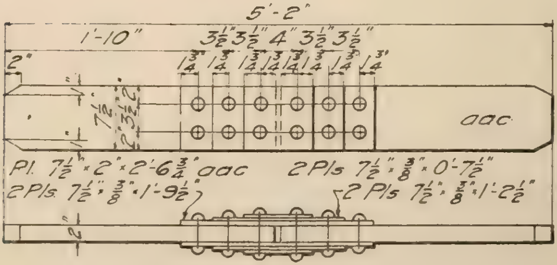

TB 19, 20, 21, 22.

FIG. 5. RIVETED JOINT TEST PIECES.

TABLE 4.

CLASSIFICATION OF TEST JOINTS.

All rivets were ⅞ inch in diameter. All surfaces in contact were painted with one coat of graphite paint. Joints designated as shop-riveted were riveted with a hydro-pneumatic riveting machine; those designated as field-riveted joints were riveted by a hand pneumatic hammer.

Joint	Material of Plates and Rivets	Number of Joints Tested	Method of Riveting	Number of Rivets Sheared	Single or Double Shear	Cross section of Main Plates inches	Filler Plates Each Side	Remarks
TB1S	Ni. Steel	3	Shop	1	Single	6½x⅜	None	Tested in tension
TB1F	“	2	Field	1	“	“	“	“
TB1L	Cr.Ni.Steel	3	Shop	1	“	“	“	“
TB2S	Ni. Steel	3	Shop	2	Single	6½x⅜	None	Do
TB2F	“	2	Field	2	“	“	“	“
TB2L	Cr.Ni.Steel	3	Shop	2	“	“	“	“
TB3S	Ni. Steel	3	Shop	3	Single	6½x⅜	None	Do
TB3F	“	2	Field	3	“	“	“	“
TB3L	Cr.Ni.Steel	3	Shop	3	“	“	“	“
TB4S	Ni. Steel	3	Shop	1	Double	6½x¾	None	Do
TB4F	“	2	Field	1	“	“	“	“
TB4L	Cr.Ni.Steel	3	Shop	1	“	“	“	“
TB5S	Ni. Steel	3	Shop	2	Double	6½x¾	None	Do
TB5F	“	2	Field	2	“	“	“	“
TB5L	Cr.Ni.Steel	3	Shop	2	“	“	“	“
TB6S	Ni. Steel	3	Shop	3	Double	6½x¾	None	Do
TB6F	“	2	Field	3	“	“	“	“
TB6L	Cr.Ni.Steel	3	Shop	3	“	“	“	“
TB7S	Ni. Steel	3	Shop	3	Double	6½x¾	One	Do
TB7F	“	2	Field	3	“	“	“	“
TB7L	Cr.Ni.Steel	3	Shop	3	“	“	“	“
TB8S	Ni. Steel	3	Shop	3	Double	6½x¾	One	Tested in tension Filler anchored with 1 rivet
TB8F	“	2	Field	3	“	“	“	
TB8L	Cr.Ni.Steel	3	Shop	3	“	“	“	
TB9S	Ni. Steel	3	Shop	3	Double	6½x¾	One	Tested in tension Filler anchored with 2 rivets
TB9F	“	2	Field	3	“	“	“	
TB9L	Cr.Ni.Steel	3	Shop	3	“	“	“	
TB10S	Ni. Steel	3	Shop	3	Double	6½x¾	Two	Tested in tension
TB10F	“	2	Field	3	“	“	“	“
TB10L	Cr.Ni.Steel	3	Shop	3	“	“	“	“
TB11S	Ni. Steel	3	Shop	3	Double	6½x¾	Two	Tested in tension Fillers anchored with 1 rivet
TB11F	“	2	Field	3	“	“	“	
TB11L	Cr.Ni.Steel	3	Shop	3	“	“	“	
TB12S	Ni. Steel	3	Shop	3	Double	6½x¾	Two	Tested in tension Fillers anchored with 2 rivets
TB12F	“	2	Field	3	“	“	“	
TB12L	Cr.Ni.Steel	3	Shop	3	“	“	“	
TB13S	Ni. Steel	3	Shop	6	Double	7½x¾*	None	Tested in tension
TB13F	“	2	Field	6	“	“ *	“	“
TB13L	Cr.Ni.Steel	3	Shop	6	“	“ *	“	“
TB14S	Ni. Steel	3	Shop	6	Double	7½x¾*	One	Do
TB14F	“	2	Field	6	“	“ *	“	“
TB14L	Cr.Ni.Steel	3	Shop	6	“	“ *	“	“
TB15S	Ni. Steel	3	Shop	6	Double	7½x¾*	One	Tested in tension Fillers anchored with 1 rivet
TB15F	“	2	Field	6	“	“ *	“	
TB15L	Cr.Ni.Steel	3	Shop	6	“	“ *	“	
TB16S	Ni. Steel	3	Shop	6	Double	7½x¾*	One	Tested in tension Fillers anchored with 2 rivets
TB16F	“	2	Field	6	“	“ *	“	
TB16L	Cr.Ni.Steel	3	Shop	6	“	“ *	“	

* Double plates.

TABLE 4.—(*Continued.*)

Joint	Material of Plates and Rivets	Number of Joints Tested	Method of Riveting	Number of Rivets Sheared	Single or Double Shear	Cross Section of Main Plates inches	Filler Plates Each Side	Remarks
TB17S	Ni Steel	3	Shop	6	Double	7½x¾*	Two	Tested in tension Fillers anchored with 2 rivets
TB17F	"	2	Field	6	"	" *	"	
TB17L	Cr.Ni.Steel	3	Shop	6	"	" *	"	
TB18S	Ni. Steel	3	Shop	6	Double	7½x¾*	Two	Tested in tension Fillers anchored with 3 rivets
TB18F	"	2	Field	6	"	" *	"	
TB18L	Cr.Ni.Steel	3	Shop	6	"	" *	"	
TB19S	Ni. Steel	2	Shop	6	Double	7½x2	None	For tests in alternate tension and compression
TB20S	"	2	"	6	"	"	"	
TB21S	"	2	"	6	"	"	"	
TB22S	"	2	"	6	"	"	"	
TB19F	Ni. Steel	2	Field	6	Double	7½x2	None	Do.
TB20F	"	2	"	6	"	"	"	"
TB21F	"	2	"	6	"	"	"	"
TB22F	"	2	"	6	"	"	"	"
TB19L	Cr.Ni.Steel	4	Shop	6	Double	7½x2	None	Do.
TB20L	"	4	"	6	"	"	"	"
TB21L	"	4	"	6	"	"	"	"
TB22L	"	4	"	6	"	"	"	"

* Double plates.

Table 4 gives a classification of the test joints with a tabulation of their principal dimensions. In all 176 joints were tested. For the nickel-steel joints five or more specimens of each type were tested and of each type some were riveted with a hydro-pneumatic riveter and others were riveted with a hand pneumatic hammer. Joints riveted with the hydro-pneumatic riveter were marked S and are hereafter referred to as shop-riveted joints. Joints riveted with a hand pneumatic hammer were marked F and are hereafter referred to as field-riveted joints. All chrome-nickel-steel joints were riveted with a hydro-pneumatic riveter. They were marked L. In all test joints the surfaces in contact had been given one coat of graphite paint.

Only one type of joint was tested under alternate tension and compression (see Fig. 5). There were thirty-two joints of this type.

7. *Testing Machine and Its Arrangement.*—The tension tests were made in the 600 000-lb. screw-power Riehle testing machine of the Laboratory of Applied Mechanics of the University of Illinois. The ends of the specimens were held in the heads of the machine by wedge-shaped grips. In testing, the load was applied at a speed of head of machine of 0.4 in. per min.

The thirty-two joints designed to withstand alternate tension and compression were also tested in the 600 000-lb. testing machine. In the tests in alternate tension and compression, the machine was fitted with an auxiliary cross-head and with two auxiliary nuts on the screws. Fig. 6 shows in diagram the arrangement of apparatus, and Fig. 7 shows the general appearance of the machine fitted with the auxiliary apparatus.

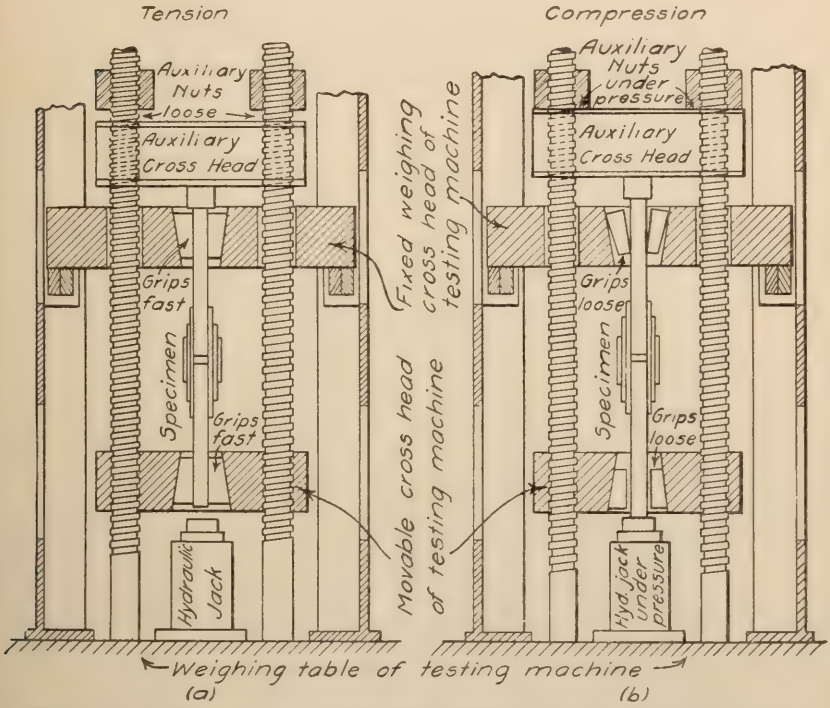

Fig. 6. Auxiliary Apparatus for Tests in Alternate Tension and Compression.

Tension and compression were applied alternately as follows: To put the specimen in tension the load was applied through the screws of the machine and weighed in the usual manner. During this part of the test the auxiliary cross-head above the weighing head of the testing machine rested on the top of the specimen, and the auxiliary nuts were clear of the auxiliary cross-head and turned with the screws. (See Fig. 6 (a)). The specimen, of course, was held by the wedge grips. The tension load was next released, after which the lower grips loosened themselves from the specimen. The hydraulic jack under the specimen was then pumped up, and its rising plunger carried the specimen and the auxiliary cross-head upward until the auxiliary cross-head was forced against the auxiliary nuts and the specimen put in compression. Fig. 6 (b) shows this condition. The reaction of the hydraulic jack on the weighing table was weighed by balancing the beam of the testing machine, and, of course, was equal to the compression load on the specimen. The pressure was next released from the jack, the weight of the auxiliary cross-head acted to tighten the grips in the weighing head of the machine, the lower grips were tightened by hand, and another tension load was applied. It will be seen that the load in compression was as accurately determined as that in tension.

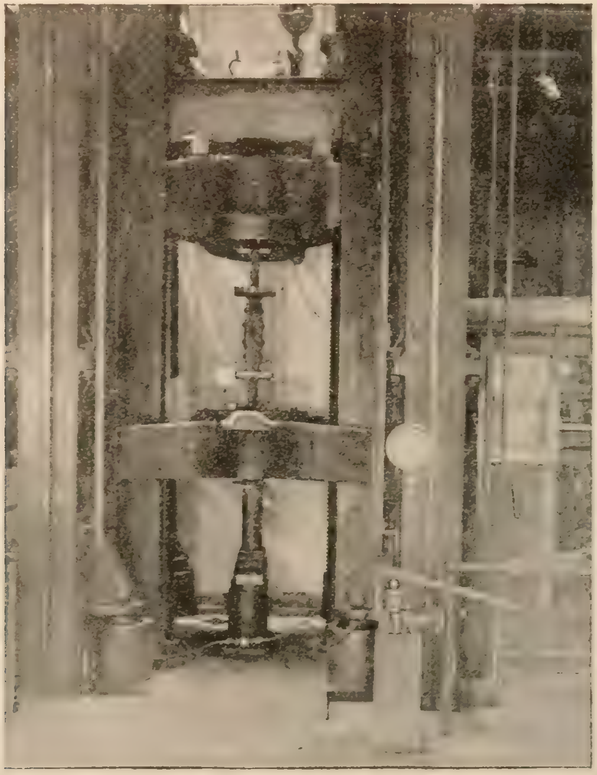

FIG. 7. VIEW OF TESTING MACHINE WITH AUXILIARY APPARATUS FOR TESTS IN ALTERNATE TENSION AND COMPRESSION.

8. *Measurement of Deformation.*—In the tension tests the elongation of the specimen was measured by means of the apparatus shown in Fig. 8. This instrument consists of two frames AA_1 which are fastened to the specimen by pointed screws SS_1, these screws being symmetrically placed with respect to the longitudinal axis of the specimen. As the specimen stretches, the frames tend to separate, but at the right hand side (as shown in the figure) they are held at a constant distance apart by means of the rod R, whose ends receive the pivots FF, and by the springs XX. At the left hand side the lower frame carries a dial D around which rotates a pointer. On the axle of this pointer is a small sheave one inch in circumference, which is driven by a fine insulated copper wire wrapped around it. This wire, which is kept taut by the weight W, is attached at its upper end to the pin P which is carried at the left hand side of the upper frame. This wire and the axis of the rod R are equidistant from the axis of

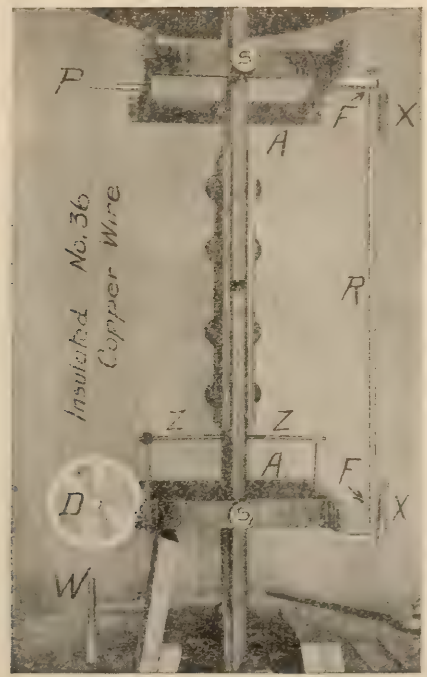

Fig. 8. View of Extensometer Attached to Test Piece.

the specimen, and thus as the specimen stretches, the pin P and the dial D separate by twice the stretch of the specimen, and this double stretch is indicated by the pointer on the dial. The dial was graduated to measure the stretch of the piece to one ten-thousandth of an inch. Before making tests with this apparatus, the dial wrapped with wire was tested in comparison with a screw micrometer, and it was found under repeated reversals to be free from slip of wire over drum and free from appreciable error. The apparatus behaved with entire satisfaction during the test and, it is believed, gave thoroughly trustworthy results.

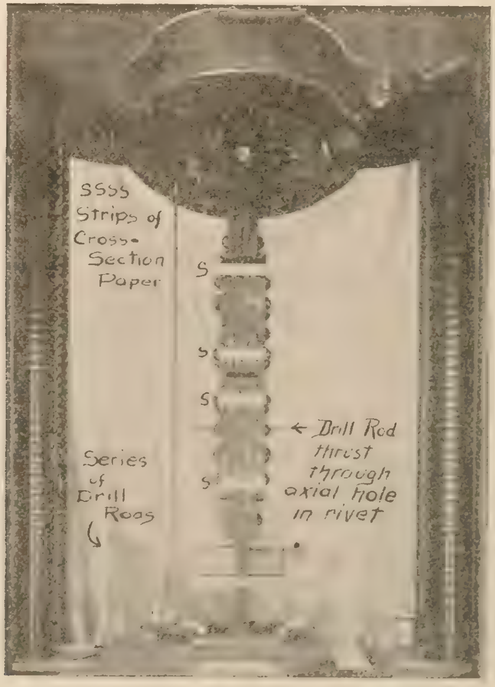
SSSS Strips of Cross-Section Paper
S
S
S
S
← Drill Rod thrust through axial hole in rivet
Series of Drill Rods

FIG. 9. VIEW OF TENSION TEST PIECE WITH AUXILIARY APPARATUS FOR MEASURING DEFORMATION.

For the purpose of measuring the movement of the several plates of a joint, strips of cross-section paper were glued across the narrow sides of the plates before testing the joints, and after the glue had hardened, these strips were cut into pieces along the edges of the plates, leaving a piece attached to each plate. As the plates slipped over each other the amount of slip was read directly from the cross-section paper. Fig. 9 and 10 show this arrangement.

In the tests in alternate tension and compression the same apparatus was used to measure the deformation of the joints as was used in the tension tests.

In one or more joints of each type the bending of the rivets was investigated. The apparatus used for detecting the bending of rivets is shown in Fig. 9 and 11. Holes 0.204 inch in diameter were drilled through all rivets of the joint as nearly axially as possible. A series of Stubs' steel drill rods varying in diameter from 0.180 inch to 0.204 inch by steps of about 0.003 inch was provided, and under the initial load the largest drill rod which could be thrust through the hole was determined by trial. This arrangement is illustrated in Fig. 11 (a) where d_1 is the diameter of this drill rod. Under any other load on the joint, if appreciable bending of the rivet had occurred, this drill rod could not be thrus

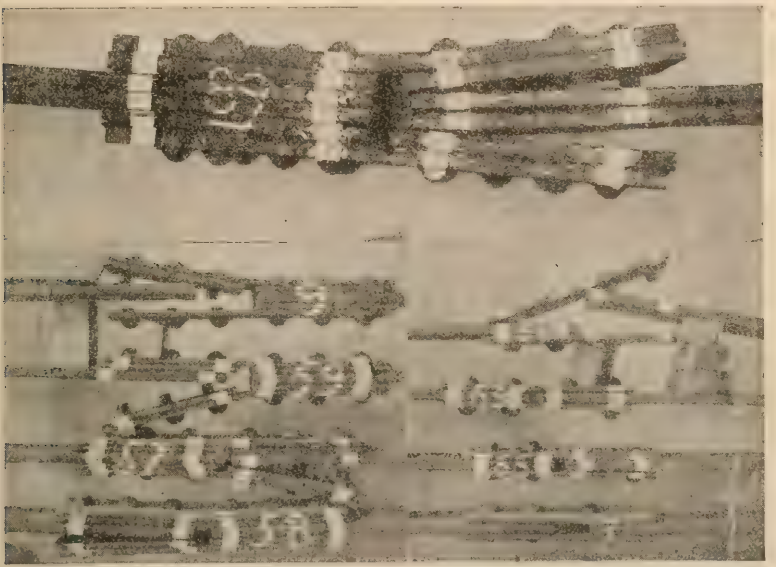

FIG. 10. VIEW OF FRACTURED TEST PIECES SHOWING MOVEMENT OF PLATES.

through the hole in the rivet, and the maximum size of rod which would pass the hole was of smaller diameter (d_2), as illustrated in Fig. 11 (b). $d_1 - d_2$ gives the total amount of bending and detrusion under the given load.

9. *Procedure of Tests.*—In the tension tests an initial load of 1000 lb. was first applied, and the dial of the extensometer set to zero; the first desired load was then applied and the dial read; the load was then reduced to the initial load of 1000 lb. and the dial again read; the next higher load was then applied, and the load again reduced to the initial load. This process was repeated until a load equal to about three-quarters of the estimated breaking load had been applied. At this point the measuring instrument was removed, and in the further test the load was progressively applied to failure, the reading of the cross-section paper being taken at intervals throughout the entire test.

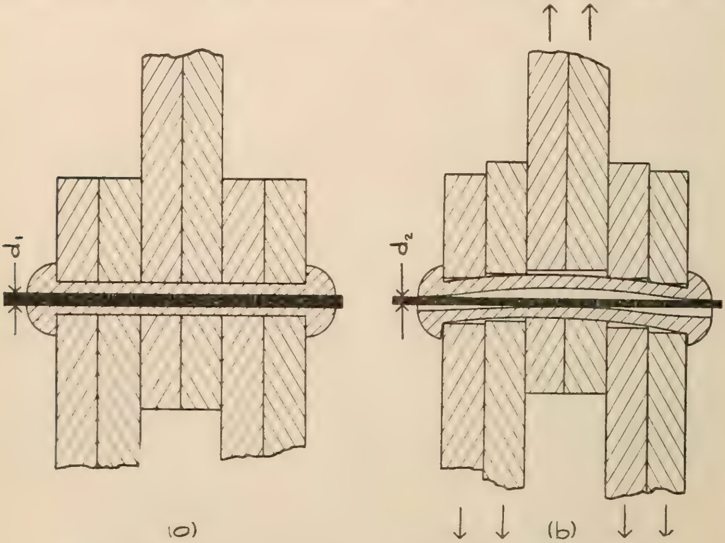

FIG. 11. DIAGRAM SHOWING METHOD OF MEASURING BENDING OF RIVETS.

In the tests of the joints designed to withstand alternate tension and compression (sixteen joints of nickel-steel and sixteen joints of chrome-nickel-steel of the form and dimensions shown in Fig. 5), the following schedule was carried out:

(a) Six specimens tested in tension, with release of load and observation of deformation and set as the load was progressively increased. (Joints TB20L, TB21L, TB20L, TB20S, TB21S, and TB22F).

(b) Six specimens tested in compression, with release of load and observations of deformation and set as the load was progressively increased. (Joints TB19L, TB20L, TB22L, TB22S, TB20F, and TB22F).

(c) Six specimens tested in alternate tension and compression, a tension load being first applied followed by an equal load in compression, then an increased load applied in tension followed by an equal load in compression, etc., the joint being finally broken in tension. (Joints TB22L, TB22L, TB22L, TB20F, TB21S, and TB19S).

(d) Four specimens tested as in (c), and in addition the bending of the rivets was investigated by the aid of holes drilled through them. (Joints TB21L, TB20L, TB21F, TB19F).

(e) Four specimens tested as in (c), except that the compression load applied was only one-half of the preceding tension load. (Joints TB19L, TB19L, TB20S, and TB22S).

(f) Two specimens tested as in (c), except that in all but the higher loads the cycle of tension and compression for any given load was repeated five times before the next larger load was applied. (Joints TB19L and TB20S).

(g) Four specimens tested by repeatedly applying alternately in tension and compression a load producing a rivet shear of about 10 000 lb. per sq. in., and occasionally interjecting a single application of a higher load. (Joints TB21L, TB21L, TB19F, and TB21F).

During the application and release of a load, throughout the above schedule, observations of deformation were taken at several loads, in order that the stress-deformation diagram of the specimen might be plotted during the application of tension, the release of tension, the application of compression, and the release of compression, i. e., throughout the complete cycle of stress.

In all tests in the second series except tests 69, 70, 105 and 106 the increment of load was 25 000 lb. In nearly all cases the maximum load applied before removing the measuring instruments was 300 000 lb. After the removal of the instruments a higher load was applied and the piece was broken. In tests 69, 70, 105 and 106 ((g) in the schedule of tests) the method of procedure was as follows: The specimen was subjected to ten cycles of alternated loading, the load both in tension and compression being such as to produce a shearing stress of about 10 000 lb. per sq. in.

in the rivets. The specimen was then subjected to one cycle of stress at loads which gave a shearing stress of about 15 000 lb. per sq. in. in rivets. Then followed ten cycles with 10 000 lb. per sq. in. shear in rivets, then one cycle with 20 000 lb. per sq. in., then ten cycles with 10 000 lb. per sq. in., then one cycle with 25 000 lb. per sq. in., and finally one cycle with 10 000 lb. per sq. in. After this the piece was broken in tension.

III. Data and Discussion.

10. *Sample Data of Tension Tests.*—Table 5, page 25, gives the data of a sample tension test of the riveted joints, (Joint TB10S, Test 83). In this particular test the bending of the rivets was investigated by means of the device already described. The data are self-explanatory. Fig. 12 shows the plotted data of this test. The lines drawn solid show deformation under stress and the lines drawn broken show permanent set after removal of stress. Complete data for all tests and curves for each test piece are on file in the Laboratory of Applied Mechanics of the University of Illinois, but in general only summarized data will be given in this bulletin.

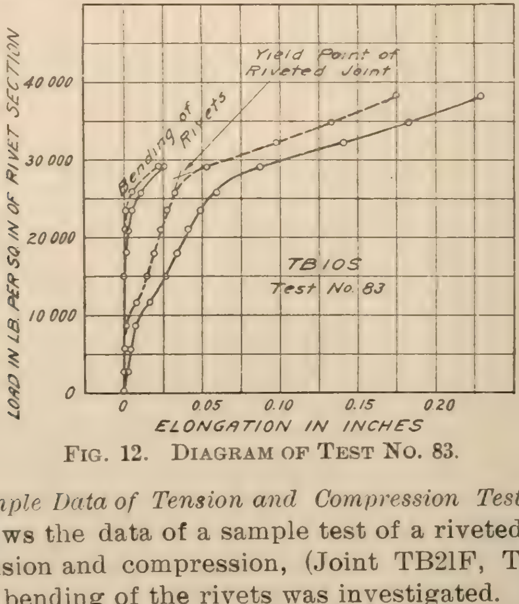

FIG. 12. DIAGRAM OF TEST NO. 83.

11. *Sample Data of Tension and Compression Tests.*—Table 6, page 27, shows the data of a sample test of a riveted joint under alternate tension and compression, (Joint TB21F, Test 94). In this test the bending of the rivets was investigated. Fig. 13 has been plotted from the data. In this test loads in tension were

followed by equal loads in compression. Fig. 14 shows the plotted results of a test in which loads in tension were followed by half as great loads in compression. Fig. 15 (Joint TB21L) shows the plotted results of a test under stress in tension only; a diagram of a test in compression only would be quite similar in appearance.

TABLE 5.

Sample Data of Tension Test of Riveted Joint.

Shear per square inch of rivet is based on the shearing area computed from the nominal diameter of rivet—⅞ inch.

Specimen TB10S. Test No. 83.

Load pounds	Shear per square in. of rivet pounds	Extensometer Gauge Length 30 in.		Drill Number of Largest Rod through Rivet. Holes 0.204 in. in Diameter.					
		Elongation inches	Set inches						
1 000	290		0.0000	8	7	8	8	8	8
9 600	2 810	.0025		8	7	8	8	8	8
1 000	290		.0000	8	7	8	8	8	8
19 500	5 710	.0042		8	7	8	8	8	8
1 000	290		.0005	8	7	8	8	8	8
30 200	8 820	.0075		8	7	8	8	8	8
1 000	290		.0015	8	7	8	8	8	8
40 000	11 720	.0168		8	7	8	8	8	8
1 000	290		.0079	8	7	8	8	8	8
51 200	15 000	.0260		8	7	8	8	8	8
1 000	290		.0146	8	7	8	8	8	8
61 500	18 000	.0337		8	8	8	8	8	8
1 000	290		.0192	8	7	8	8	8	8
71 800	21 040	.0409		9	8	8	8	8	8
1 000	290		.0234	8	8	8	8	8	8
80 400	23 550	.0495		9	8	9	9	8	9
1 000	290		.0272	8	8	8	8	8	8
87 700	25 680	.0588		11	11	11	9	9	11
1 000	290		.0325	9	9	9	8	8	9
99 600	29 160	.0875					13	12	14
1 000	290		.0531	14	13	14	11	11	12
109 800	32 150	.1410							
1 000	290		.0978						
119 400	34 920	.1823							
1 000	290		.1335						
128 800	37 700	.2286							
1 000	290		.1750						
182 200	53 320	Maximum load							

Upper rivets on north side sheared between second filler and main plate.

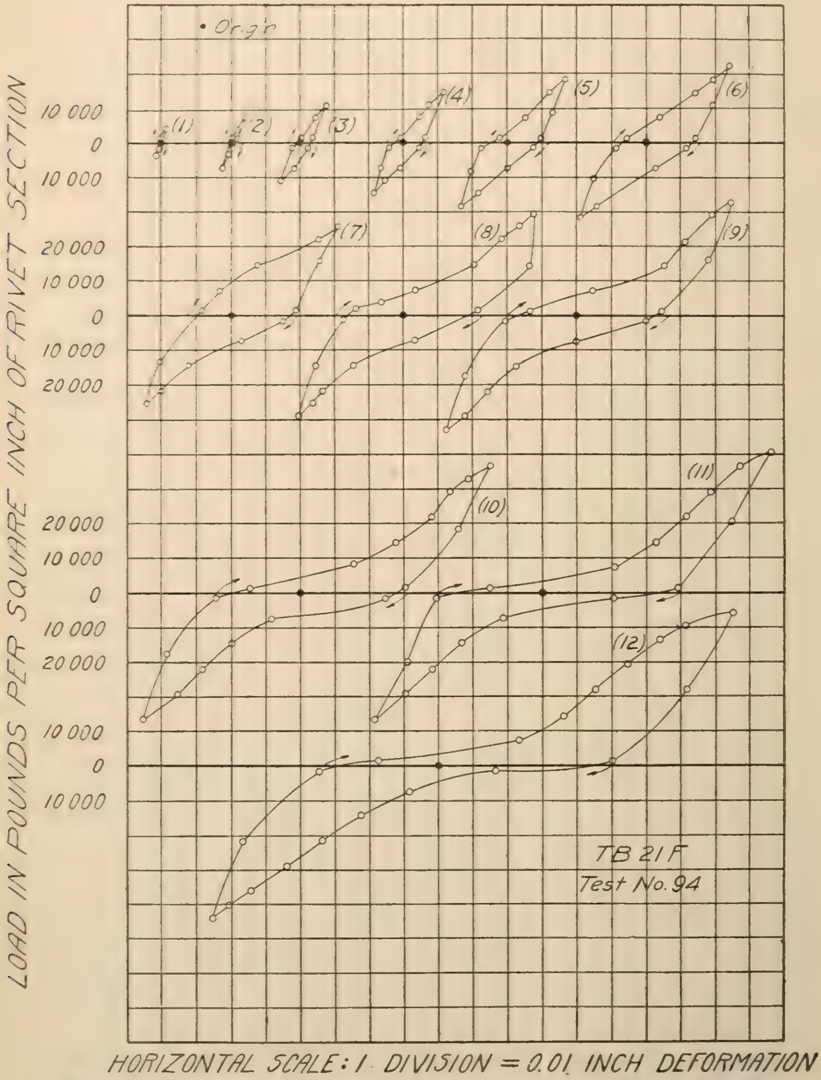

FIG. 13. DIAGRAM OF TENSION AND COMPRESSION TEST NO. 94.

TABLE 6.

SAMPLE DATA OF TEST OF RIVETED JOINTS IN ALTERNATE TENSION AND COMPRESSION.

T denotes load in tension; C, load in compression. Shear per square inch of rivet is based on the shearing area computed from the nominal diameter of rivet—⅞ inch.

Specimen TB21F. Test No. 94.

Load pounds	Shear per Square Inch of Rivet. pounds	Elongation in 20 Inches	Drill Number of Largest Rod through Rivet. Holes 0.204 in. in Diameter
T 7 000	1 025	+.0000	6 6 6 6 6 6
T 28 000	4 100	+.0014	6 6 6 6 6 6
T 7 000	1 025	+.0007	
C 10 000	1 465	—.0007	
C 25 000	3 665	—.0014	
C 10 000	1 465	—.0007	
T 7 000	1 025	+.0003	
T 25 000	3 665	+.0016	
T 50 000	7 330	+.0041	6 6 6 6 6 6
T 7 000	1 025	+.0020	
C 10 000	1 465	+.0005	
C 25 000	3 665	—.0005	
C 50 000	7 330	—.0025	6 6 6 6 6 6
C 10 000	1 465	—.0009	
T 7 000	1 025	+.0004	
T 50 000	7 330	+.0044	
T 75 000	10 990	+.0075	
T 7 000	1 025	+.0037	
C 10 000	1 465	+.0021	
C 50 000	7 330	—.0016	
C 75 000	10 990	—.0054	6 6 6 6 6 6
C 10 000	1 465	—.0023	
T 7 000	1 025	— .0008	
T 50 000	7 330	+.0047	
T 75 000	10 990	+.0075	
T 100 000	14 650	+.0118	6 6 6 6 6 6
T 7 000	1 025	+.0062	
C 10 000	1 465	+.0044	
C 50 000	7 330	—.0010	
C 75 000	10 990	—.0052	
C 100 000	14 650	—.0088	6 6 6 6 6 6
C 51 000	7 475	—.0068	
C 10 000	1 465	—.0042	
T 7 000	1 025	—.0026	
T 50 000	7 330	+.0050	
T 100 000	14 650	+.0120	
T 124 500	18 230	+.0167	6 6 6 6 6 6
T 60 000	8 790	+.0130	
T 7 000	1 025	+.0094	
C 10 000	1 465	+.0072	
C 50 000	7 330	—.0002	
C 100 000	14 650	—.0089	
C 125 000	17 320	—.0140	6 6 6 6 6 6
C 57 000	8 355	—.0108	
C 10 000	1 465	—.0078	
T 7 000	1 025	—.0058	
T 50 000	7 330	+.0038	
T 100 000	14 650	+.0142	
T 125 000	18 320	+.0195	
T 152 500	22 350	+.0242	6 6 6 6 6 6
T 75 000	10 990	+.0194	
T 7 000	1 025	+.0141	
C 10 000	1 465	+ 0115	
C 50 000	7 330	+.0024	
C 100 000	14 650	—.0112	
C 125 000	18 320	—.0147	
C 150 000	21 980	—.0192	6 6 6 6 6 6
C 70 000	10 260	—.0155	
C 10 000	1 465	—.0109	
0 000	0 000	—.0080	
T 7 000	1 025	—.0087	
T 50 000	7 330	+.0035	
T 100 000	14 650	+.0075	
T 150 000	21 980	+.0250	
T 176 000	25 785	+.0307	6 6 6 6 6 6
T 108 000	15 830	+.0255	
T 7 000	1 025	+.0179	
C 10 000	1 465	+.0148	
C 50 000	7 330	+.0029	
C 100 000	14 650	—.0125	
C 150 000	21 980	—.0202	
C 175 000	25 640	—.0246	6 6 6 6 6 6
C 91 000	13 340	—.0210	
T 10 000	1 465	—.0143	
T 25 000	3 665	—.0065	
T 50 000	7 330	+.0035	
T 100 000	14 650	+.0202	
T 150 000	21 980	+.0283	
T 175 000	25 640	+.0235	
T 200 500	29 370	+.0374	6 6 7 7 7 6
T 100 000	14 650	+.0309	
T 7 000	1 025	+.0213	6 6 6 6 6 6
C 10 000	1 465	+.0178	
C 50 000	7 330	+.0032	
C 100 000	14 650	—.0148	
C 150 000	21 980	—.0237	
C 175 000	25 640	—.0262	
C 200 000	29 300	— .0303	7 6 7 7 7 7
C 101 000	14 800	—.0256	
C 10 000	1 465	—.0173	6 6 6 6 6 6
T 7 000	1 025	—.0132	
T 50 000	7 330	—.0048	
T 100 000	14 650	—.0245	
T 150 000	21 980	—.0319	
T 200 000	29 300	--.0395	
T 224 000	32 820	—.0450	6 7 8 8 8 8
T 110 000	16 120	—.0385	
T 7 000	1 025	+.0248	
C 10 000	1 465	+.0205	
C 50 000	7 330	+.0000	
C 100 000	14 650	—.0174	
C 150 000	21 980	—.0253	
C 200 000	29 300	—.0322	
C 225 000	32 960	—.0372	7 6 7 7 7 7
C 120 000	17 580	—.0321	
C 10 000	1 465	—.0206	7 6 7 6 6 6
T 7 000	1 025	—.0147	
T 50 000	7 330	+.0152	
T 100 000	14 650	+.0278	
T 150 000	21 980	+.0380	
T 200 000	29 300	+.0432	
T 225 000	32 960	+.0488	
T 251 000	36 770	+.0552	6 8 9 9 8 8
T 125 000	18 320	+.0458	
T 7 000	1 025	+.0302	6 7 8 8 7 7
C 10 000	1 465	+.0248	
C 50 000	7 330	—.0085	
C 100 000	14 650	—.0198	

TABLE 6. (*Continued.*)

SAMPLE OF DATA OF TEST OF RIVETED JOINTS IN ALTERNATE TENSION AND COMPRESSION.

T denotes load in tension; C, load in compression. Shear per square inch of rivet is based on the shearing area computed from the nominal diameter of rivet—⅞ inch.

Specimen TB21F. Test No. 94.

Load pounds	Shear per Square Inch of Rivet. pounds	Elongation in 20 Inches	Drill Number of Largest Rod through Rivet. Holes 0.204 in. in Diameter
C 150 000*	21 980	−.0283	
C 200 000	29 300	−.0352	
C 250 000	36 630	−.0393	
C 250 000	36 630	−.0455	8 7 8 8 8 8
C 121 000	17 730	−.0386	
C 10 000	1 465	−.0243	7 7 7 7 7 7
T 0 000	0 000	−.0152	
T 50 000	7 330	+.0208	
T 100 000	14 650	+.0330	
T 150 000	21 980	+.0415	
T 200 000	29 300	+.0489	
T 250 000	36 630	+.0575	
T 275 000	40 300	+.0666	7 9 11 9 11 9
T 140 000	20 500	+.0550	
T 7 000	1 025	+.0392	6 8 8 8 7 7
C 10 000	1 465	+.0208	
C 50 000	7 330	−.0112	
C 100 000	14 650	−.0235	
C 150 000	21 980	−.0320	
C 200 000	29 300	−.0398	
C 250 000	36 630	−.0588	
C 275 000	40 300	−.0572	9 8 11 9 11 8
C 137 000	20 070	−.0492	
C 10 000	1 465	−.0308	8 8 8 8 8 8
T 7 000	1 025	−.0175	
T 50 000	7 330	−.0235	
T 100 000	14 650	+.0362	
T 150 000	21 980	+.0458	
T 200 000	29 300	+.0548	
T 250 000	36 630	+.0642	
T 275 000	40 300	+.0720	
T 300 000	43 960	+.0852	8 12 13 13 13 12
T 150 000	21 980	+.0722	
T 7 000	1 025	+.0502	7 9 11 11 11 9
C 10 000	1 465	+.0168	
C 50 000	7 330	−.0085	
C 100 000	14 650	−.0225	
C 150 000	21 980	−.0338	
C 200 000	29 300	−.0437	
C 250 000	36 630	−.0542	
C 275 000	40 300	−.0605	
C 300 000	43 960	−.0747	12 11 14 13 14 11
C 150 000	21 980	−.0667	
C 10 000	1 465	−.0441	11 9 11 9 11 8
T 392 800	57 550		

Failed by shearing 6 upper rivets on both sides.

*Slipped about .0100.

In four tests an effort was made to study the behavior of the test pieces under several reversals of a load which gave a shearing stress of about 10 000 lb. per sq. in. of rivet area. In Fig. 16, page 38, in the test of each riveted joint the diagram marked (1) shows the deformation of the specimen under the first cycle of load (i.e., during the application of tension, the release of tension, the application of compression, and the release of compression); diagram marked (10) shows the deformation during the tenth cycle of load; (11) shows the deformation during a cycle in which the rivet shear was increased to 15 000 lb. per sq. in.; (12) shows the next cycle which was for a load of 10 000 lb. per sq. in. rivet shear, etc.

12. *Tables and Diagrams of Results.*—Table 7 gives the shearing stress in the rivets (stated in lb. per sq. in. of the shearing area of the rivet) at which the first noticeable slip of the riveted

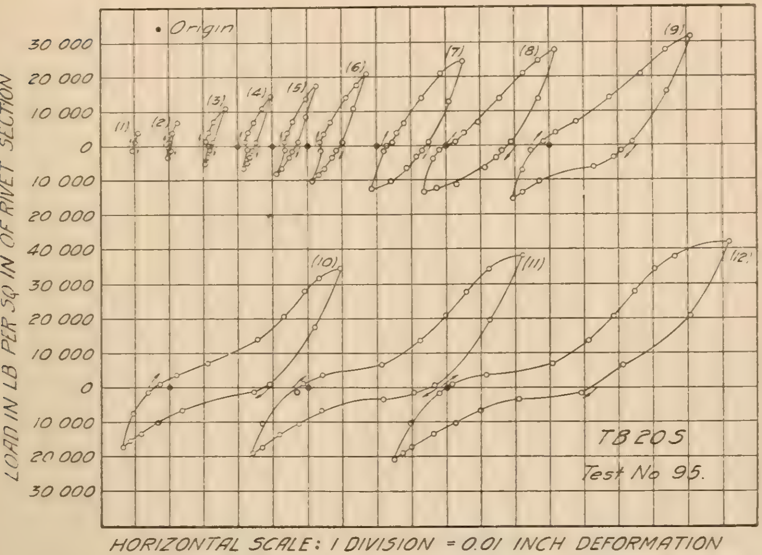

FIG. 14. DIAGRAM OF TENSION AND COMPRESSION TEST NO. 95.

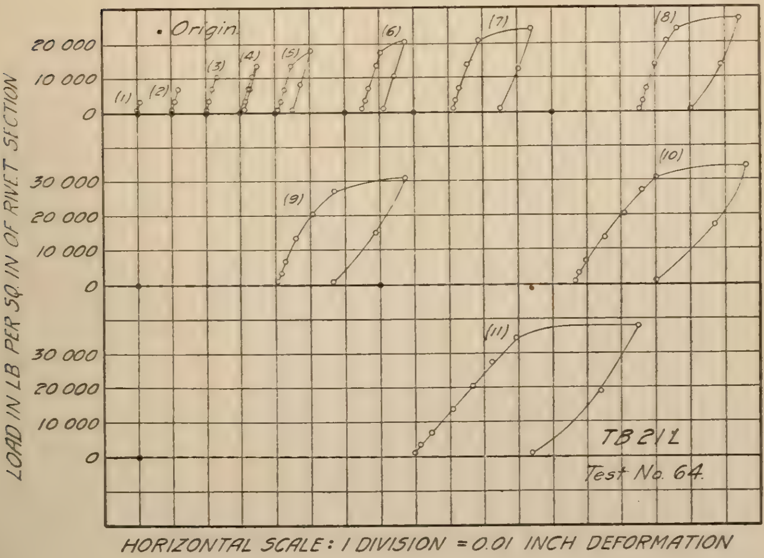

FIG. 15. DIAGRAM OF TENSION TEST NO. 64.

TABLE 7.

FIRST NOTICEABLE SLIP OF RIVETED JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Test piece	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Nickel steel Riveted Joints.																		
Shop riveted	8 900 10 500 11 000	10 500 6 300 11 000	8 200 5 600 15 000	7 500 6 900 7 600	6 100 8 000	6 000 6 000 5 900	10 000 9 000 8 800	9 500 10 000 10 500	11 600 12 200 10 500	12 500 9 000 11 800	11 500 12 500 13 000	13 000 14 000 12 500	9 500 12 000 8 600	11 500 10 000 10 500	15 000 11 000 14 100	11 000 17 500 13 000	14 000 14 500 12 500	15 500 18 000 10 500
Field riveted	10 500 17 000	17 500 13 000	15 000 10 500	5 200 10 000	5 500 5 800	5 600 5 600	8 000 7 500	6 600 9 700	8 800 7 500	8 600 7 500	9 100 9 000	8 500 7 500	11 100 8 400	12 500 8 500	11 400 11 000	9 500 9 500	13 000 11 000	15 000 14 000
Av. shop riveted	10 130	9 270	9 600	7 330	7 050	5 970	9 270	10 000	11 430	11 100	12 330	13 000	10 030	10 670	13 370	13 830	13 800	14 670
Av. field riveted	10 750	15 250	12 750	7 605	5 650	5 600	7 750	8 150	8 150	8 050	9 050	8 000	9 750	10 500	11 200	9 500	12 000	14 500
Av. shop and field riveted	10 380	11 660	10 860	7 440	6 350	5 820	8 660	9 260	10 120	9 880	11 020	11 100	9 920	10 600	12 500	12 080	13 080	14 600
General average 10 300																		
Chrome-nickel-steel Riveted Joints.																		
	13 000 15 000 15 000	13 000 14 000 13 500	11 000 13 500 11 000	11 500 10 000 10 000	12 000 12 000 13 000	10 500 11 500 10 000	12 000 12 000	14 500 14 000 13 500	14 000 13 000 11 500	13 500 13 500 11 500	16 500 14 000 14 000	16 500 15 000 15 000	16 500 16 500 14 000	15 000 16 500 12 500	16 000 18 000 10 000	15 000 15 000 16 000	15 000 13 500 19 000	16 500 16 500 13 500
Average	14 330	13 500	11 830	10 500	12 330	10 670	12 000	14 000	12 830	12 830	14 830	15 500	15 670	14 670	14 670	15 330	15 830	15 500
General average 13 690																		
Carbon-steel Riveted Joints. (Maintenance of Way Association Tests).																		
	27 000 17 000 18 000 20 000 19 000	13 000 14 000 14 500 16 500 13 000	8 000 13 500 10 500 12 500 12 500	16 000 15 000 11 500 11 500 12 500	14 000 12 500 7 500 7 000 11 500	6 500 7 000 7 500 5 500 5 500	6 500 6 500 7 500 7 000 7 500	7 500 8 000 7 000 7 000	12 000 13 500 12 000 11 000 9 000	6 000 8 000 7 000 6 000 7 500	6 000 6 000 9 000 10 500 9 500	8 000 10 000 11 000 13 500 11 000	9 000 8 500 9 000 10 000 8 000	6 000 6 500 7 000 7 000 7 500	8 500 5 500 6 000 6 000 5 000	6 500 6 500 4 000 8 500 6 000	5 000 4 500 5 000 6 500 6 000	5 500 6 000 7 000 5 500 6 500
Average	20 200	14 200	11 400	13 300	10 500	6 400	7 000	7 380	11 500	6 900	8 200	10 700	8 900	6 800	6 200	6 300	5 400	6 100
General average 9 310																		

joint occurred in the tension tests. In all cases in computing the area of a rivet the nominal diameter of the rivet was used. The shearing stress is taken at the load at which the total apparent movement amounted to about 0.0025 in. The determination of this point involves the use of some judgment, but as the same method was used in all cases the results are considered to be comparable with each other.

Table 8, page 32, gives the yield point of the nickel-steel and the chrome-nickel-steel riveted joints stated in terms of the shearing stress on the rivets.

In Fig. 17 and 18 (at the end of the text) permanent set is plotted for the nickel-steel riveted joints, the chrome-nickel-steel riveted joints, and the carbon-steel riveted joints. In each case the results given are the average for the test pieces of the set. A general comparison may be made by eye.

Table 9, page 34, gives the ultimate strength of the nickel-steel and of the chrome-nickel-steel riveted joints stated in lb. per sq. in. of the shearing area of the rivets. In computing rivet area the nominal diameter was used. The ultimate strength of the carbon-steel joints reported by the Maintenance of Way Association is also given.

Table 10 gives the ultimate shearing strength and Table 11, the yield point of riveted joints tested in alternate tension and compression.

Fig. 19 to 22 (at the end of the text) show graphically the total deformation produced by different intensities of stress. In the direct tension and direct compression tests, the set is also plotted, and in the other tests except tests 69, 70, 105, and 106, both the total range of deformation and the deformation under tension are given. Table 11 gives the yield point of the joints tested in alternate tension and compression.

13. *Slip in Tension Tests.*—By the use of delicate extensometers slip of the rivets in a test joint can be detected and measured under small loads. An inspection of Fig. 17 and 18 shows that the plates began to slip at loads as low as the usual working loads, the amount of slip increasing regularly until the yield point of the joint was reached. The first noticeable slip of the riveted joint given in Table 7 is taken when the total apparent movement amounted to 0.0025 inch. The slip of joint in the Maintenance of Way tests of carbon-steel riveted joints is also included in Table 7.

TABLE 8.

YIELD POINT OF RIVETED JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Test Piece	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Nickel-steel Riveted Joints																		
Shop riveted	37 800 35 500 36 600*	36 200 35 500 34 500*	30 600 32 100 35 500*	35 000 34 100 34 900*	30 100 35 100 34 100*	31 600 37 000 33 300*	31 400 32 700 31 400*	36 200 35 500 35 700*	36 800 37 700 38 300*	29 900 29 200 27 800*	34 900 37 600 36 200	40 100 37 500 38 200*	35 000 35 000 31 300*	28 900 28 300 27 500*	32 300 31 900 33 700	34 000 35 600 35 200	28 800 27 600 27 000	31 800 32 500 30 000
Field riveted	37 000 35 500	35 800 38 000	34 300 34 000	37 000 34 500	34 000 36 800	35 700 35 100	34 000 32 500	36 800 37 000	36 200 37 200	30 200 30 000	35 300 34 200*	36 800 37 000	40 000 36 100	29 200 31 300	31 100 32 700*	34 500 35 000	27 000 28 000	31 700 29 800
Av. shop riveted	36 630	35 400	32 730	34 670	33 100	33 970	31 830	35 800	37 600	28 970	36 230	38 600	34 770	28 230	32 630	34 930	27 800	31 430
Av. field riveted	36 250	36 900	34 150	35 750	35 400	35 400	33 250	36 900	36 700	30 100	34 750	36 900	38 050	30 250	33 400	34 750	27 500	30 750
Average shop and field riveted	36 480	36 000	33 300	35 100	34 020	34 540	32 400	36 240	37 240	29 420	35 640	37 920	36 080	29 040	32 910	34 860	27 680	31 160
Yield point of rivet in bending	38 600	32 400	32 500	35 000	32 600	29 000	31 000	32 700	37 400	23 500	35 600	32 100	31 500	22 000	28 100			
Chrome-nickel-steel Riveted Joints.																		
	34 500	36 800	36 500	35 000	32 000	36 000	31 300	34 000	37 000	24 500	29 500	34 500	32 300	25 500	28 100	31 000	23 400	27 800
	37 200	34 000	34 500	36 000	36 000	34 700	29 800	33 800	35 700	24 000	30 200	30 900	30 600	25 000	30 500	32 200	24 100	27 500
	38 000	37 500	35 500	39 200	36 200	37 300	32 000	31 300	34 600	24 400	28 700	31 100	30 600	25 000	28 000	30 000	25 000	27 000
Average	36 570	36 100	35 500	36 730	34 730	36 000	31 030	33 030	35 770	24 300	29 470	32 170	31 170	25 170	28 870	31 070	24 170	27 430
Yield point of rivet in bending	36 300	36 000	33 600	38 000	35 000	36 700	30 900	32 000	35 200	24 400	29 000	32 800	30 600	25 500	29 100	31 500	24 200	27 100

* Joint with hole drilled through rivet.

TABLE 8.—(*Continued.*)

YIELD POINT OF RIVETED JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Test Piece	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Carbon-nickel-steel Riveted Joints—(Maintenance of Way Association Tests).																		
	28 000 31 400 29 900 31 000 32 800	24 900 27 100 25 500 27 000 26 300	26 100 26 400 27 100 25 200 24 100	28 800 27 000 28 000 27 700 26 000	29 600 27 600 25 100 26 800 29 300	24 700 23 000 23 000 21 600 24 100	20 100 19 300 19 400 21 900 22 800	26 600 25 000 27 400 26 200 28 000	26 100 25 800 23 800 27 800 27 900	19 400 19 500 19 300 19 400 19 400	25 000 23 300 25 100 19 800 22 300	29 500 27 500 26 000 27 800 26 000	28 500 28 000 27 800 28 900 27 900	18 600 20 600 18 900 18 700 18 800	20 700 19 800 19 600 21 700 20 000	21 200 17 700 21 500 22 300 22 000	15 800 16 000 15 400 16 000 16 000	17 600 16 800 17 600 16 600 17 000
Average	30 620	26 160	25 780	27 500	27 680	23 280	20 700	26 610	26 280	19 400	23 100	27 360	28 220	19 120	20 360	20 940	15 840	17 120
Excess of nickel steel over carbon steel, per cent	19 2	37.6	29.6	27.6	23.0	48.4	56.6	36.4	41.8	51.8	54.4	38.7	27.9	52.0	61.9	66.5	75.0	82.0
Excess of chrome nickel steel over carbon steel, per cent	19.4	37.7	37 6	33.5	25.5	51.7	50.0	21.0	36.1	25.4	27.6	17.6	10.3	31.6	41.8	48.3	53.0	60.2

TABLE 9.

ULTIMATE SHEARING STRENGTH OF RIVETED JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Test piece	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Nickel-steel Riveted Joints.																		
Shop riveted.	63 900 58 900 60 600*	58 600 58 080 58 300*	55 000 55 600 57 500*	57 700 56 700 55 800*	46 950 56 600 55 800*	56 500 56 700 54 800*	55 900 56 100 55 500*	57 700 58 900 58 200*	57 700 57 600 58 400*	55 100 54 300 53 300*	55 200 58 000 57 600	59 700 56 400 57 500*	55 700 56 600 57 800*	51 400 53 000 52 300*	53 700 54 200 55 000	53 700 55 300 56 900	55 000 56 600 54 500	56 000 55 400 55 300
Field riveted	60 000 57 300	56 900 58 500	56 300 56 200	59 400 59 000	58 300 58 000	58 400 58 400	57 200 57 600	59 300 59 300	58 100 57 600	55 000 56 100	57 700 57 300*	59 700 57 700	57 500 57 400	52 400 53 100	57 200 56 600*	56 700 56 700	56 200 53 400	55 300 54 600
Av. shop riveted	61 130	58 300	56 030	56 730°	53 100	56 000	55 830	58 270	57 900	54 230	56 930	57 870	56 700	52 230	54 300	55 300	55 400	55 570
Av. field riveted	58 650	57 700	56 250	59 200	58 150	58 400	57 400	59 300	57 850	55 550	57 500	58 700	57 450	52 750	56 900	56 700	54 400	54 950
Av. shop and field riveted	60 140	58 080	56 120	57 720	55 120	56 960	56 460	58 460	57 880	54 760	57 160	58 200	57 000	52 440	55 340	55 860	55 140	55 320
Max. tensile stress in plate (lb. per sq. in.)	48 150	33 300	48 950	46 850	33 100	49 700	49 100	49 600	49 700	47 800	49 400	50 800	48 400	45 300	47 900	47 600	47 400	46 800
Chrome-nickel-steel Riveted Joints.																		
	53 350 56 400 56 900	57 100 54 250 57 650	55 700 53 850 55 750	54 550 57 300 58 100	52 700 55 650 56 550	53 940 54 500 56 300	51 300 51 450 53 100	55 000 53 250 51 600	53 950 55 200 53 620	48 900 48 150 47 530	51 550 53 420 50 600	54 800 51 520 50 720	52 450 53 000 52 050	48 400 48 300 49 000	51 350 51 550 50 450	51 950 52 800 51 280	51 150 49 620 50 650	51 320 50 900 50 920
Average	55 550	56 330	55 100	56 650	54 970	54 910	51 950	53 280	54 260	48 190	51 860	52 350	52 500	48 570	51 120	52 010	50 470	51 050
Max. tensile stress in plate (lb. per sq. in.)	46 170	32 750	49 500°	46 510	32 200	48 000°	45 250°	46 840°	45 950°	41 650°	45 550°	46 650°	44 350	41 000	43 150	44 180	42 800	42 950

* Joints with hole drilled through rivets. ° Plate after rupture showed "stress lines" indicating that the yield point of the plate material had been passed.

TABLE 9.—(*Continued.*)

ULTIMATE SHEARING STRENGTH OF RIVET JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Test piece	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Carbon-steel Riveted Joints (Maintenance of Way Association Tests).																		
	53 200	47 600	45 700	53 500	50 100	48 500	50 000	48 800	48 800	46 300	48 800	53 900	48 800	46 900	49 200	48 400	45 400	45 500
	36 600	48 400	44 900	50 700	50 400	45 300	47 800	48 900	48 900	50 800	44 000	48 400	54 700	46 600	48 000	45 000	44 600	46 300
	54 900	48 200	45 500	52 100	50 600	47 400	45 700	49 500	48 800	45 000	50 100	51 400	49 200	44 000	46 800	45 900	44 300	47 300
	54 500	48 100	47 120	52 400	50 900	47 900	50 900	50 400	50 400	52 100	51 400	48 200	48 500	47 700	48 100	50 000	45 500	46 300
	55 700	49 100	45 700	51 600	52 400	45 500	49 200	49 900	56 800	47 500	50 800	47 600	51 500	45 100	46 800	48 500	44 900	46 200
Average	50 980	48 280	45 780	52 060	50 880	46 920	48 720	49 500	50 740	48 340	49 020	49 900	50 540	46 060	47 780	47 760	44 940	46 320
Max. tensile stress in plate (lb. per sq. in.)	15 800	27 900	40 100	15 190	29 800	41 300	43 300	42 900	48 400	44 300	43 700	45 900	45 840	39 900	41 200	41 800	37 100	39 600
Excess of nickel-steel over carbon steel in shearing stress (per cent)	17.2	20.4	22.7	10.9	8.4	21.4	13.9	18.2	13.2	13.3	16.5	16.7	12.9	14.0	15.9	17.2	22.6	19.4
General average	16.4																	
Excess of chrome-nickel-steel over carbon steel (per cent)	9.1	16.6	20.5	8.9	8.2	17.1	6.6	7.7	7.0	−1.5	5.6	5.1	4.0	5.3	7.0	9.1	12.5	10.4
General average	8.85																	

TABLE 10.

ALTERNATE TENSION AND COMPRESSION TESTS. ULTIMATE SHEARING STRENGTH OF RIVETED JOINTS.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Nickel-steel Riveted Joints			Chrome-nickel-steel Riveted Joints		
Joint	Load at Failure lb. per sq in.	Remarks	Joint	Load at Failure lb. per sq. in.	Remarks
Riveted Joints Subjected to Tension only.					
TB20S	56 700		TB20L	52 400	
TB21S	58 100		TB20L	49 800	
TB22F	58 800		TB21L	49 800	
Av.	57 870		Av.	50 670	
Riveted Joints Subjected to Compression only.					
TB20F		End of plates butted.	TB19L		Joints buckled.
TB22S	55 600	Rivets sheared.	TB20L		Plates butted.
TB22F		Joint buckled.	TB22L		
Riveted Joints Subjected to Tension followed by Half as Great Compression.					
TB20S	57 400		TB19L	50 200	
TB22S	55 200		TB19L	49 700	
Av	56 300			49 950	
Riveted Joints Subjected to Tension followed by Equal Compression.					
TB19S	56 200		TB19L	49 500	
TB20S	56 600		TB20L*	49 300	
TB21S	55 820		TB21L*	51 700	
			TB21L°	50 800	
Av. shop riveted	56 200		TB21L°	51 600	
			TB22L	51 600	
			TB22L	52 000	
TB19F*	58 260		TB22L	51 700	
TB19F°	57 500				
TB20F	57 700		Av.	51 020	
TB21F*	57 550				
TB21F°	60 250				
Av. field riveted	58 230				
Av. field and shop riveted	57 480				

* Riveted joints with holes drilled through rivets.

° Riveted joints subjected to cycles of reversed load causing a shearing stress in the rivets of 10 000 lb. per sq. in.

TABLE 11.

Alternate Tension and Compression Tests. Yield Point of Riveted Joints.

Loads are given in terms of the stress in pounds per square inch on the nominal shearing area of the rivets.

Nickel-steel Riveted Joints				Chrome-nickel-steel Riveted Joints			
Joint	Load at Yield Point lb. per sq. in.	Range of Deformation at Yield Point		Joint	Load at Yield Point lb. per sq. in	Range of Deformation at Yield Point	
		Under Load inches	Set inches			Under Load inches	Set inches
Riveted Joints Subjected to Tension only.							
TB20S	34 000	0.050	0.030	TB20L	29 500*	0.067	0.047
TB21S	34 600	0.045	0.028	TB20L	32 000	0.027	0.020
TB22F	36 100	0.041	0.028	TB21L	23 000*	0.040	0.027
Av.	34 900	0.045	0.029	Av.	28 170	0.045	0.032
Riveted Joints Subjected to Compression only.							
TB20F	37 000	0.035	0.020	TB19L	18 000*		
TB22S	33 700	0.045	0.030	TB20L	30 500	0.040	0.030
TB22F	36 200	0.035	0.020	TB22L	31 200	0.035	0.025
Av.	35 630	0.038	0.023	Av.	30 850§	0.037	0.027
Riveted Joints Subjected to Tension followed by Half as Great Compression.							
TB20S	39 000	0.079		TB19L	20 700*	0.015	
TB22S	38 900	0.095		TB19L	24 300*	0.017	
Av.	38 950	0.087		Av.	22 500	0.016	
Riveted Joints Subjected to Tension followed by Equal Compression.							
TB19S	33 800	0.089		TB19L	17 200*	0.015	
TB19F†	36 600	0.085		TB20L†	15 700*	0.017	
TB20F	36 400	0.092		TB21L†	15 300*	0.013	
TB20S	34 800	0.095		TB22L	19 500*	0.027	
TB21S	31 800	0.088		TB22L	16 000*	0.015	
TB21F†	36 500	0.100		TB22L	15 000*	0.015	
Av.	34 970	0.091		Av.	16 450	0.017	
Riveted Joints Subjected to Cycles of Reversed Load Causing a Shearing Stress in the Rivets of 10 000 lb. per sq. in.							
TB19F	30 300	0.045‡		TB21L	24 100	0.050‡	
TB21F	32 700	0.052‡		TB21L	25 900	0.055‡	
Av.	31 500	0.049		Av.	25 000	0.052	

* Yield point not well defined.

† Riveted joints with holes drilled through rivets.

‡ Range in deformation in tension.

§ TB19L omitted in taking average.

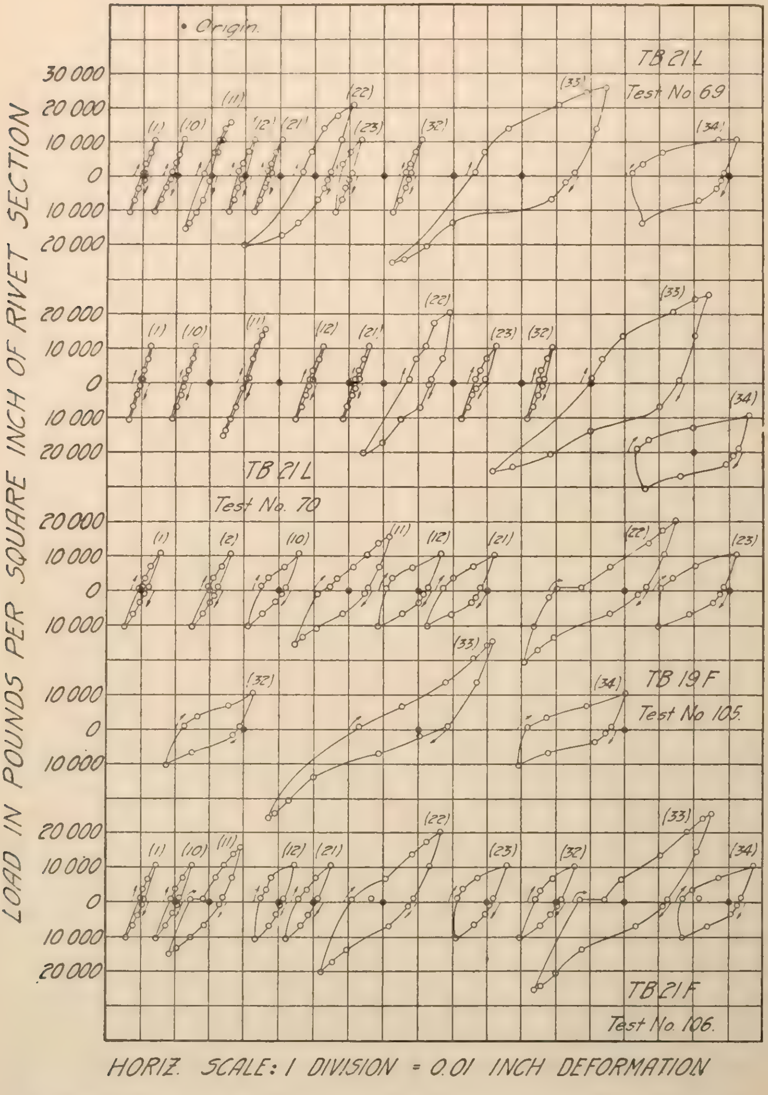

FIG. 16. DIAGRAM OF TENSION AND COMPRESSION TESTS UNDER ALTERNATIONS OF LOW STRESSES.

In general, the nickel-steel riveted joints showed first noticeable slip under a slightly higher load than that producing first noticeable slip in the carbon-steel joints of the Maintenance of Way Association. It may be doubted whether a comparison of the general averages would be significant. If the first slip of the joint is caused by the overcoming of the friction of the rivet heads and plates, no reason is apparent for expecting a higher load at first slip in nickel-steel riveted joints than in carbon-steel riveted joints, and the higher resistance of the nickel-steel riveted joints as compared with the carbon-steel riveted joints may be due to differences in fabrication.

In the tests of chrome-nickel-steel riveted joints, both plates and rivets were made up of steel softer and weaker than that used in the nickel-steel riveted joints, but the first noticeable slip took place at a load considerably higher than the load causing first slip in either the nickel-steel joints or the carbon-steel joints, indicating that the strength of material has very little effect on the stress at first slip of joint.

14. *Yield Point in Tension Tests.*—The term "yield point of riveted joints" will be used to designate the point at which a marked increase of yield in the riveted joint occurs and at which also a marked set of the riveted joint is found. As the loads are increased beyond the load causing a first noticeable slip, the permanent set increases regularly for several increments of load, as if the rivets were gradually seating themselves against the sides of the rivet holes; then quite suddenly the set increases markedly and the slope of the set curve becomes noticeably less. The method of determining the yield point in the tension tests is shown in Fig. 12. Some judgment must be exercised in determining this point, but it is quite definite except for the joints with long rivets. The increased deformation seems to be due partly to bending of the rivet and partly to shearing detrusion. Before the yield point is reached, a considerable movement of the riveted joint as a whole has taken place, and, of course, beyond the yield point the deformation increases rapidly. As measured by the displacement of the paper strips, the amount of slip up to the yield point in the riveted joints with short rivets and without filler plates was about 0.025 in., and after rupture the slip of the unbroken end was about 0.20 in. After the yield

point of the joint had been passed, the movement was due in greater part to the shearing detrusion of the rivet. In the joints with long rivets, a greater movement occurred, as much as 0.05 in. before the yield point was reached, and after rupture as much as 0.40 in. in the unbroken end of several of the riveted joints.

15. *Ultimate Strength in Tension Tests.*—All the riveted joints tested, both nickel-steel and chrome-nickel-steel, finally failed by shearing of rivets. The ultimate shearing stress in the rivets is given in Table 9. Whether a riveted joint will fail by shear of rivets or otherwise depends upon the design of the joint and the relative strength of the rivet and plate material. A fact not always recognized is that the stress at which a rivet will break under shear depends not only upon the quality of the rivet material but also on the relative hardness of the rivet and of the plate on which the rivet bears. The rivet material used in the nickel-steel riveted joints was about 16 per cent stronger in tension than that used in the chrome-nickel-steel riveted joints; the plate material of the nickel-steel riveted joints was considerably stronger and harder than the rivet material, while in the chrome-nickel-steel riveted joints the plate material was not much stronger than the rivet material; the result was the shearing strength of the nickel-steel riveted joints was only 7 per cent greater than that of the chrome-nickel-steel riveted joints, about half of the gain which might be expected from a comparison of the tests of rivet material.

Comparing the ultimate strength of the Maintenance of Way Association tests of carbon-steel riveted joints, the apparent gain in ultimate strength is only about 16 per cent for the nickel steel and about 9 per cent for the chrome nickel steel. However, the carbon-steel riveted joints were designed so as to have nearly the same strength in shear of rivets and in tension in plates; this is shown by the fact that some failed by shear of rivets and some by tension in plates. The tensile stresses in the plates forming the riveted joints, developed at the ultimate load, are given in Table 9. In the case of the nickel-steel riveted joints it will be noted by reference to the strength of the specimens cut from the plates (see Table 2) that even in the riveted joints in which three or six rivets were sheared the tensile stress in the plate at rupture of the riveted joint was in no case as great as the yield point of the material of the plates. The nickel-steel and the chrome-nickel-steel riveted joints were made with the same dimensions as the Maintenance of Way Association riveted joints and as they all failed by shear of rivets and as no scaling or other indication of exceeding the yield point of the plates was observed in the nickel-steel riveted joints, and no chrome-nickel-steel riveted joints failed in the plates, (though many of the plates in the chrome-nickel-steel riveted joints were stressed beyond the yield points as was shown by "scaling" and by the appearance of characteristic "stress lines",) it is evident that the efficiency of joint was less than in the carbon-steel joints, and that a larger gain could have been shown for the nickel-steel and chrome-nickel-steel riveted joints had they been proportioned so as to have as high joint efficiencies as had the Maintenance of Way Association carbon-steel joints. The tests, therefore, may not be used to compare the full strength of riveted joints of the three classes of materials.

In connection with this topic the results of the tests made by Preuss at Darmstadt are of interest (see foot note on page 5). In the test joints of that series nearly all the rivets were of nickel steel. The plate material was not specified, and Dr. Preuss writes that its nature is not known; it probably was ordinary structural steel. At any rate, judging from the rivet material, it seems probable that the rivets of the joints were of stronger and harder material than were the plates. Comparing the butt joints in Preuss' tests with those most nearly similar in the University of Illinois tests, the ultimate shearing stress in the rivets of Preuss' test joints was about 19 per cent greater than the ultimate shearing stress in the rivets of the corresponding Illinois test joints. In view of the fact that in the University of Illinois tests the plate material was stronger and harder than the rivet material, while in Preuss' tests the reverse is true, the difference in shearing strength does not seem excessive. Making similar comparisons for lap joints in the two series of tests, Preuss' results for ultimate shearing stress in rivets are about 39 per cent higher than those found at Illinois. This great difference in ultimate strength may be due in part to the fact that the riveted ends of the lap joints in Preuss' tests were bent during fabrication to make the center of the rivet come in the direct line of the pull on the joint.

16. *Slip in Alternate Tension and Compression Tests.*—The slip of the riveted joints in the alternate tension and compression tests is shown by the diagrams in Fig. 13 to 16. The most striking feature in the tests under alternate tension and compression is the very large slip which may take place under comparatively low loads. Especially is there danger of a large amount of slip if at any time a riveted joint has been subjected to a single load considerably beyond the ordinary working load. In Fig. 16, if cycle (1) is compared with cycle (12) there is seen a marked increase in the amount of slip under loads below ordinary working loads; this slip occurred in the joint after a single application of an unusually high load.

17. *Yield Point in Alternate Tension and Compression Tests.*—In the tension tests the yield point of the riveted joints was in general determined from the set curve (see Fig. 12); but in tests in alternate tension and compression there is very little set at the end of a cycle of stress and the amount of this is not significant. In these tests the yield point has been located by J. B. Johnson's method which consists in finding a point on the stress deformation curve where the rate of change of deformation is 50 per cent greater than during the early part of the test. The average yield point obtained in this way does not differ much from that obtained by taking the first marked change in direction of the stress-deformation curve. Where bending of rivet was directly examined, the first noticeable set of rivet took place at a stress not widely differing from the yield point as determined by the above method. From the results of the tests it would seem that the riveted joints tested under alternate tension and compression showed yield points about as high as those tested under stress in one direction only. Attention is called to the lower yield points found in Tests 69, 70, 105, and 106, in which the joints had been subjected to repeated applications of the load.

18. *Ultimate Strength in Alternate Tension and Compression Tests.*—Of the series of riveted joints for alternate tension and compression loads, six were tested in direct tension and six in direct compression. This was done to furnish a basis of comparison for the alternate tension and compression tests. Every joint of the series failed by shear of rivets, except that in nearly all of the direct compression tests the main plates finally came in di-

rect contact and the joints buckled noticeably, thus preventing the completion of the shearing action. In the one compression test in which the rivets were sheared to failure, the joint buckled noticeably and the failure was on the concave side of the joint. In the tension failures, in some cases rivets sheared on both sides of the main plates and in other cases on one side only. The joints failed suddenly with a loud report, the load falling off slightly just before failure. In no case was there any indication that either the main plates or the cover plates had reached their elastic limit when the ultimate load was applied.

The comparatively few alternations of load to which the nickel-steel and the chrome-nickel-steel riveted joints were subjected apparently had very little effect on the ultimate strength, as may be seen by comparing the ultimate strengths with those of the joints tested in tension. It is entirely probable, however, that a large number of reversals of load would cause failure under lower loads. Reversed load tests involving a large number of loadings on full-sized riveted joints have never been made, so far as is known. Such tests would be very expensive and would require a long time. In the absence of such tests it may be assumed that riveted joints under repeated reversals of load would be in danger of failure at loads very much lower than under steady tension loads.

19. *Bending of Rivets.*—It has long been recognized that in riveted joints the rivets are subjected to bending stresses, and attempts have been made to calculate these stresses. A mathematical analysis of the bending of a rivet is subject to many uncertainties: the ends of the rivets are partially restrained; the compression and detrusion of the rivet and plate modify the bending action and render uncertain the position of the point of application of the resulting pressure; and in various ways the conditions of beam action are indeterminate. The investigation of bending of rivets made in these tests is novel, so far as known. It shows to what extent bending takes place in such joints. The first marked bending of the rivets was found to be closely coincident with the yield point of the joint. It is seen also that the longer the rivet the greater the relative importance of the resistance to bending. It will be noted that in those joints having long rivets (TB14 to TB18) the nickel-steel riveted joints have their yield point at a stress considerably higher than do the carbon-

steel riveted joints, and that the yield point of the chrome-nickel-steel riveted joints is between the two.

In the alternate tension and compression tests, the load at which actual appreciable bending of the rivets first takes place (also generally coincident with the yield point of the joint) seems to be slightly lowered by a few applications of alternate tension and compression.

20. *Action of Riveted Joints under Steady, Repeated, and Alternated Loads.*—The tests indicate in a general way the behavior which may be expected in a riveted joint of a structure under various kinds of loading. Under a steady load in one direction, a slight slip may be expected under ordinary working loads, and the deformation of the structure will be greater than if it contained no riveted joints. Under ordinary loads the deformation due to slip of rivets will not be at all serious. If the joint were subjected to repeated applications of working load in the same direction, probably the deformation due to slip would increase but little after the first few applications of the load; in this case there would be a small permanent deformation of the structure, which would not be large enough to cause trouble, and which would increase but very little under successive loads. If the joint were subjected to ordinary working loads acting first in one direction and then in the other, the slip of rivets would doubtless be much greater than under a steady load or a repeated one-direction load of maximum amount equal to the maximum value of the alternated load. Under such stresses as are sometimes used in practice, this slip might increase under repetition of load, the joint working loose until the frictional hold of the rivet heads was greatly lessened and perhaps reduced to zero. Moreover, a single application of an overload, from accidental or other causes, might greatly increase this slip under succeeding working loads. The resulting deformation in the structure might become so great as to seriously impair its usefulness.

If a riveted joint in any structure be loaded beyond the yield point, the resulting deformation in the structure may be large enough to seriously injure it. If a riveted joint is stressed beyond the yield point under a long continued load, it might eventually fail, though data on this point are lacking. If a riveted joint is repeatedly stressed beyond the yield point by loads always acting in the same direction, the probability is that it would eventually fail. If a riveted joint is repeatedly stressed beyond the yield point by loads acting first in one direction and then in the other, the probability of failure would be great.

21. *The Basis for Design of Riveted Joints.*—It has been held that riveted joints should be designed on the basis that there will be no slip of plates at working loads and that the friction between the plates will be sufficient to sustain ordinary working loads and to prevent bearing stresses on the rivets. The experimental evidence of these tests, and of other tests already referred to shows that it would be futile to attempt to design riveted joints which shall have zero slip, whether the rivets be of carbon steel or of nickel steel. Slip must be expected below the ordinary working stresses, and if for any reason the assumed stresses are exceeded the slip of the joint will increase rapidly. It seems evident that with ordinary workmanship, the frictional hold of the plates is not an available asset of strength.

If the ultimate shearing strength of the rivets and the ultimate tensile strength of the plate are used as the basis of design, it will be important to know the shearing strength of the rivet material when used with the plate material of the riveted joints. The results of tests of shearing strength of rivet material which have been made in hardened steel dies are not directly applicable. If the shearing strength of rivet steel tested in connection with plates of the same degree of hardness be considered to be 0.80 of the tensile strength of the rivet material, it may be expected that with harder plates the ratio of shearing strength of rivet to tensile strength of rivet will be less, reaching three-quarters or even two-thirds; the ratio of shearing strength of rivet to tensile strength of plate will, of course, be still smaller. On the other hand, if the plate is softer than the rivet, the shearing strength of the rivet will be greater than otherwise. In designing riveted joints on the basis of ultimate strength, then, the shearing strength of the rivet material when used with the plate material of the riveted joint should be known. With the stronger plate material, such as nickel steel, the relatively low strength of rivet material is something of a handicap, though the higher the strength of the rivets the less this is.

Another basis which might be used is the yield point of the joint, the point at which a marked increase of yield in the riveted joint occurs. This point in the tests described seems to be coincident with the first marked increase of set of the riveted joint and generally also with the first marked bending of the rivets. In designing, this yield point of the riveted joint could be coordinated with the yield point of the plate material. The average yield point of the nickel-steel riveted joints was about 35 000 lb. per sq. in. of rivet section. The yield point of the nickel-steel plate material was 51 700 lb. per sq. in. The ratio of the yield point of riveted joint to ultimate shearing strength of riveted joint in both series of tests was about the same as the ratio of the yield point of the plate material in tension to the ultimate tensile strength of the plate material. A design of joint might be made giving equal strength for yield point of joint and yield point of net section of plate in tension. With this basis of design, it would be important to know more definitely the cause of the increase of the movement of the joint at the yield point. If it is mainly a distortion and shearing detrusion of the rivet, the hardness of the rivet material will be a governing consideration. If the bending of the rivet is of importance, the greater bending strength of the nickel-steel rivet will be of service. This feature of the action of riveted joints is worthy of further investigation.

The foregoing applies to riveted joints with the load applied in one direction only. For alternated loads, the first slip is of more importance, repetitions increase the movement, and an occasional overload makes still greater increase. Although the ultimate strength and the yield point do not seem to be seriously affected by a few alternations of load, it seems wise to keep the shearing stresses in riveted joints subjected to alternated loading so low that slip of joint will be small. For such conditions the advantage in the use of nickel-steel rivets is open to question especially if it is found, as has been claimed, that it is difficult to drive nickel-steel rivets in such way as to give effective grip on the plates they hold together.

22. *Summary.*—The following review is given:

1. A total of 90 nickel-steel riveted joints and of 54 chrome-nickel-steel riveted joints were tested in tension. These riveted joints duplicated in dimensions the series of carbon-steel riveted joints reported by the American Railway Engineering and Maintenance of Way Association. Sixteen nickel-steel riveted joints and sixteen chrome-nickel steel riveted joints were tested in tension, compression and alternate tension and compression. Stretch,

slip, and set of riveted joints were observed, and the bending of rivet was determined by means of holes drilled axially through the rivets.

2. In the tests there was a noticeable slip of joint generally at loads within ordinary working shearing stress of rivet. The movement of the joint increased fairly regularly to a load averaging about 35 000 lb. per sq. in. of rivet shear for the nickel-steel riveted joints, when a marked increase of movement was found. This increase was closely coincident with a marked set of the joint and with a marked bending of the rivet. All the riveted joints failed by shear of rivets, as was to be expected, at ultimate shearing stresses which ran fairly uniform in both the nickel-steel series and the chrome-nickel-steel series for all the types of joint tested.

3. The experimental evidence indicates that the resistance of the joint to first noticeable slip of rivet depends more upon the workmanship of the riveting than upon the quality of the rivet material, though the contractile and gripping properties of the rivets have an influence.*

4. The yield point of a riveted joint, taken as the load at which a marked increase of yield occurs, seemingly indicates a definite property of the riveted joint. This phenomenon is worthy of further investigation. The first marked bending of the rivets was found to be closely coincident with the yield point of the joint. It was found that the longer the rivet the greater the relative importance of the resistance to bending. In the alternate tension and compression tests the first appreciable bending of the rivet seemed to be slightly lowered by a few applications of load.

5. In the alternated load tests the most striking feature was the relatively large slip which took place at comparatively low loads. The amount of this slip was especially large when a riveted joint had been subjected to a single load considerably beyond the ordinary load.

---

*To determine the effect of painting the contact surfaces of riveted joints upon the load to give first noticeable slip, a phase of the subject brought out in correspondence with Mr. Albert Kingsbury, of Pittsburgh, Pennsylvania, tests for this purpose have been made since this bulletin was put in type. Joints resembling TB5 were riveted up, one set being unpainted, one set painted with graphite paint, and one set painted with red lead. All the riveted joints showed evidence of slip at loads within ordinary working shearing stress of rivets. Those painted with graphite paint gave noticeable slip at loads somewhat lower than those painted with red lead, and the unpainted test joints slipped at loads still a little higher, the differences in the three types of test joints being not large.

6. The ultimate shearing strength of riveted joints depends on the shearing strength of rivet material, and this is influenced by the relative hardness of rivets and plates.

7. The ratio of the yield point of riveted joint to ultimate shearing strength of riveted joint in the tests was about the same as the ratio of the yield point of the plate material in tension to the ultimate tensile strength of the plate material.

8. In riveted joints designed on the basis of ultimate strength, strength of rivet material and of plate material are of prime importance and the use of special steels of great strength may be of advantage.

9. In riveted joints designed on the basis of frictional hold of rivets without reference to the bending of rivets there is little advantage in using rivets of special steels of great strength since joints with such rivets show about the same resistance to first noticeable slip as do joints with ordinary carbon-steel rivets.

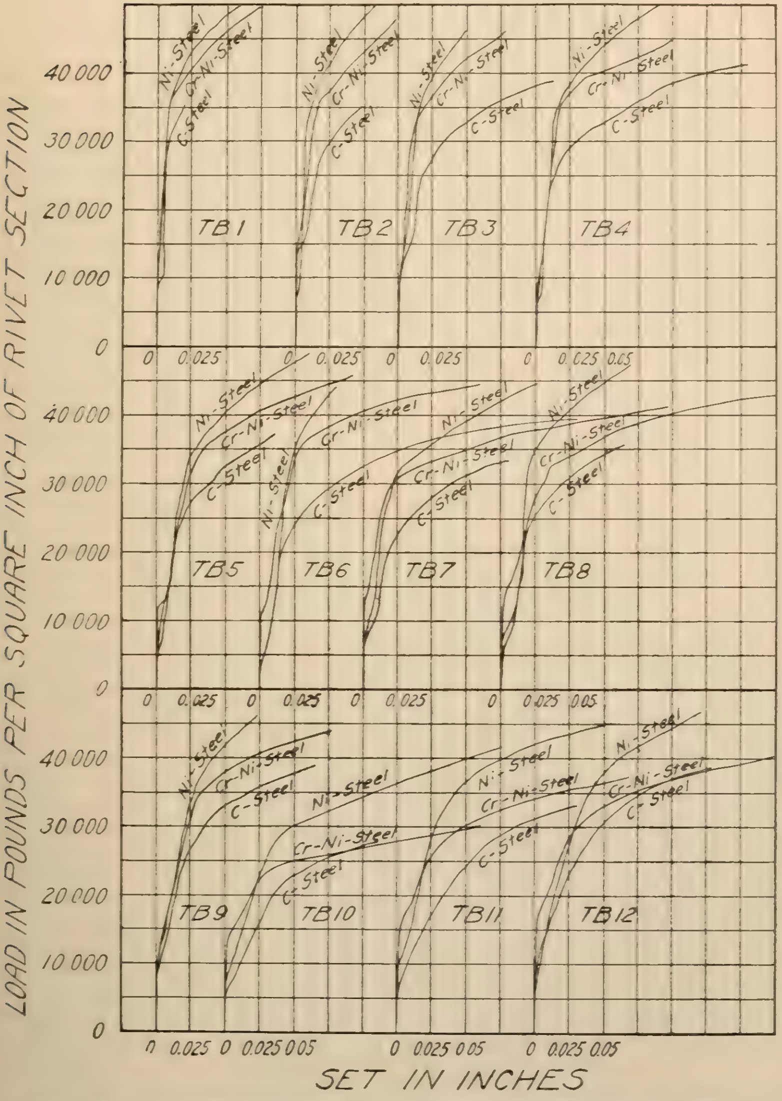

FIG. 17. DIAGRAM SHOWING PERMANENT SET IN TENSION TESTS.

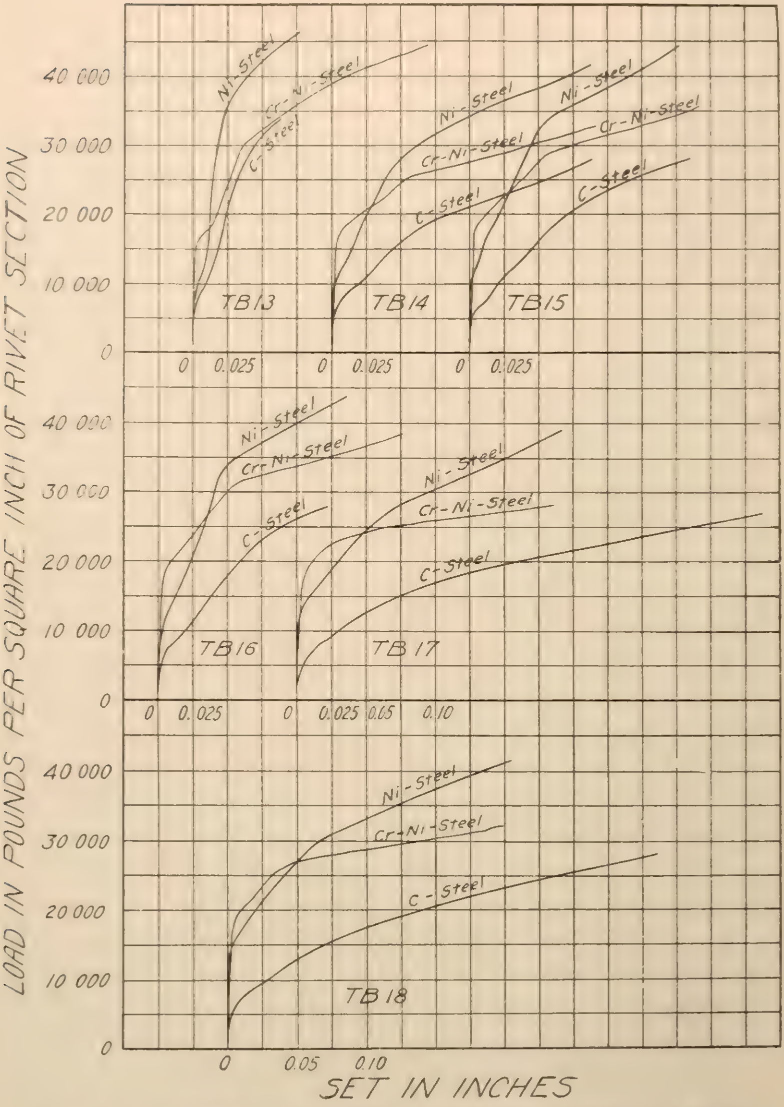

FIG. 18. DIAGRAM SHOWING PERMANENT SET IN TENSION TESTS.

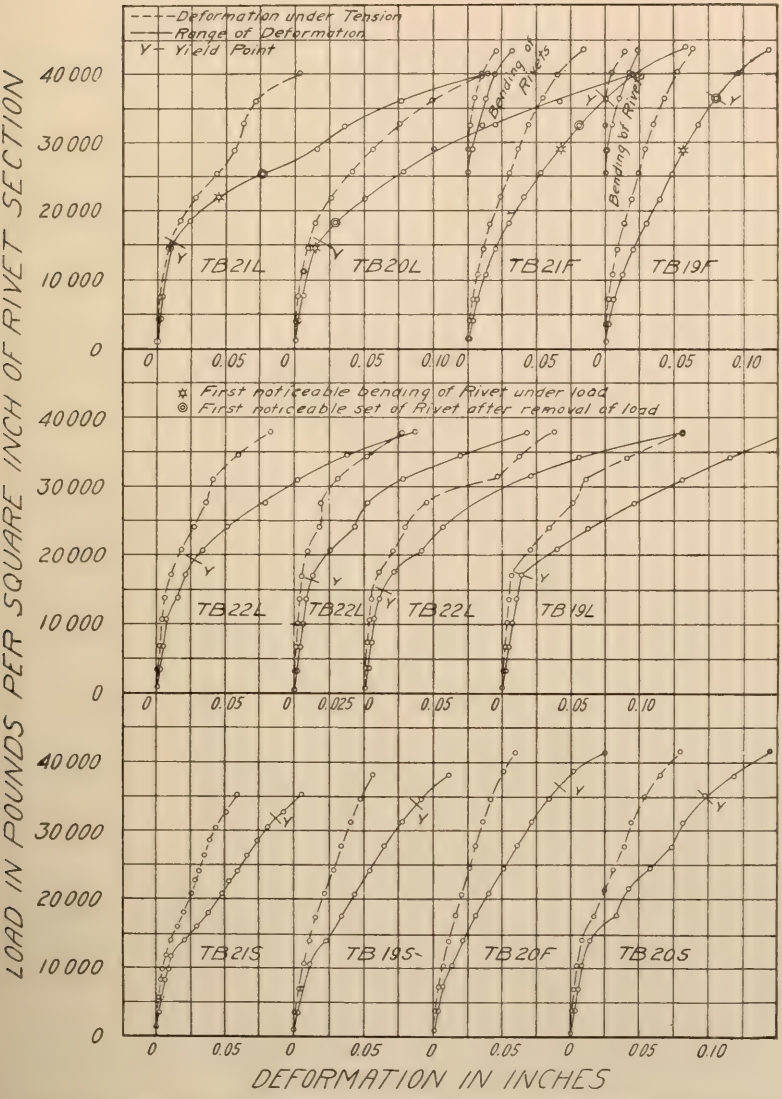

FIG. 19. DIAGRAM SHOWING DEFORMATION IN TESTS IN TENSION FOLLOWED BY EQUAL COMPRESSION.

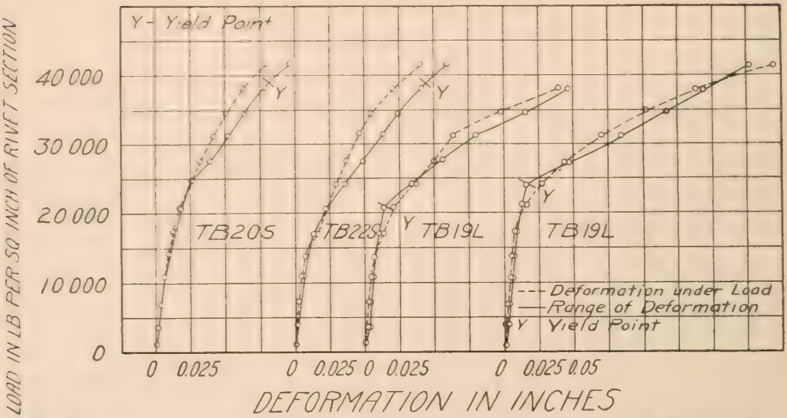

FIG 20. DIAGRAM SHOWING DEFORMATION IN TESTS IN SIMPLE TENSION AND SIMPLE COMPRESSION OF RIVETED JOINTS DESIGNED FOR ALTERNATED LOADS.

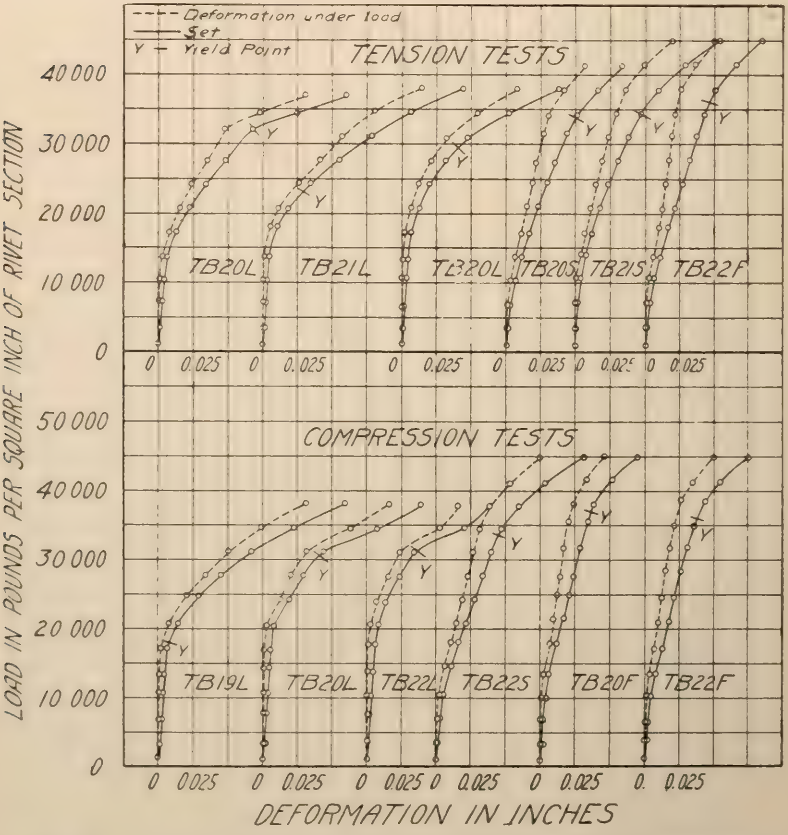

FIG. 21. DIAGRAM SHOWING DEFORMATION IN TESTS IN TENSION FOLLOWED BY HALF AS GREAT COMPRESSION.

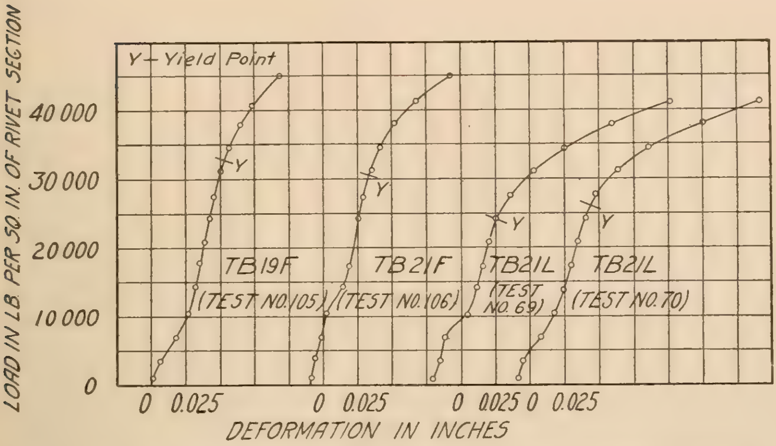

FIG. 22. DIAGRAM SHOWING DEFORMATION OF RIVETED JOINTS TESTED UNDER ALTERNATIONS OF LOW LOADS.

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*Bulletin No. 2. Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks 1905.

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*Bulletin No. 3 The Engineering Experiment Station of the University of Illinois, by L. P. Breckenridge 1906.

*Bulletin No. 4. Tests of Reinforced Concrete Beams, Series of 1905, by Arthur N. Talbot. 1906.

*Bulletin No. 5. Resistance of Tubes to Collapse, by Albert P. Carman. 1906.

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UNIVERSITY OF ILLINOIS
ENGINEERING EXPERIMENT STATION

BULLETIN NO. 50 OCTOBER, 1911

TESTS OF A SUCTION GAS PRODUCER

BY C. M. GARLAND, FORMERLY INSTRUCTOR IN MECHANICAL ENGINEERING
AND A. P. KRATZ, INSTRUCTOR IN MECHANICAL ENGINEERING

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TESTS OF A SUCTION GAS PRODUCER

I. THEORETICAL DISCUSSION

1. *Introduction.*—The chemical reactions occurring within the fuel bed of the gas producer, while well understood by the metallurgist and the gas engineer, are probably not so clear to the mass of mechanical engineers. Hence, it may be pertinent in the presentation of the results of these tests, to review briefly the theory involved in the conversion of a solid fuel into a gaseous fuel through the agency of the gas producer.

2. *Simple Carbon Monoxide Producer.*—In its simplest form, the gas producer consists of a closed retort in which carbon is burned in a limited supply of oxygen. Fig. 1 illustrates such a producer. Air enters the retort at the base, and passing up through the grate into the bed of incandescent fuel, comes in contact with the carbon in the lower zone of the fuel bed, where the oxygen of the air unites with the carbon to form CO_2. The formula expressing this reaction is:

$$C + O_2 = CO_2 \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (1)$$

Since the atomic weights of carbon and oxygen are, respectively, 12 and 16, the above formula indicates that 12 parts by weight of carbon require 32 parts by weight of oxygen and that 44 parts by weight of carbon dioxide are formed; i. e.,

$$\begin{array}{ll} & C + O_2 = CO_2 \\ & 12 + 32 = 44 \\ \text{or} & 1 + 2\tfrac{2}{3} = 3\tfrac{2}{3} \end{array}$$

This reaction is exothermic, i. e., it gives out heat, the amount of heat given out being 14 540 B. t. u. per lb. of carbon.

The CO_2 formed, remaining in contact with the incandescent carbon, begins immediately to take up more carbon, since at temperatures above 1100°F., the CO_2 becomes an oxidizing agent. This reaction is expressed by the formula:

$$CO_2 + C = 2\ CO \ldots\ldots\ldots\ldots\ldots\ldots (2)$$

which is reversible, i. e., it may take place in either direction, depending upon the temperature.

The above formula shows that 44 lb. of CO_2 unite with 12 lb.

of C and produce 56 lb. of CO. Thus:

$$CO_2 + C = 2\ CO$$
$$44 + 12 = 56$$
$$3\tfrac{2}{3} + 1 = 4\tfrac{2}{3}$$

This reaction is endothermic, i. e., it absorbs heat from the fuel bed. The amount of heat absorbed per pound of carbon taking place in the reaction is 10 100 B. t. u.

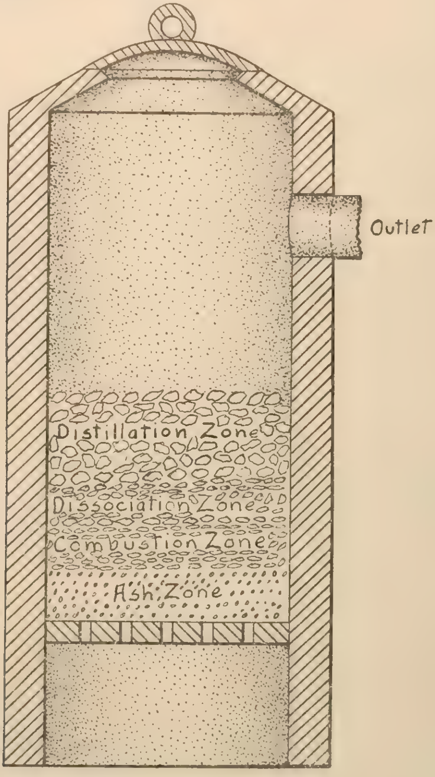

FIG. 1

As the 10 100 B. t. u. represents also the amount of heat that would be given out on combustion of the CO formed, the theoretical thermal efficiency of the carbon monoxide producer will be, therefore, if all of the carbon is assumed to be converted into carbon monoxide, $\frac{10100}{14540} = 69.5$ per cent. The remaining 30.5 per cent of the heating value of the fuel has been lost in radiation, conduction, and in sensible heat in the gases leaving the producer. The gas will consist of CO and N_2.

3. *Producer Using Water Vapor.*—In the actual commercial producer, the loss in sensible heat inherent in the simple carbon monoxide producer is reduced by utilizing this heat in the decomposition of some other agent, either the vapor of water or carbon dioxide, introduced into the fuel bed with the air. The water vapor is usually supplied by evaporating water either in an external boiler, which utilizes the sensible heat of the gases leaving the producer, or by some form of vaporizer in the form of a water jacket surrounding the fuel bed, and utilizing the heat from the fuel bed for the vaporization of the water. The steam generated in either case mixes with the air entering the producer, and passes with it into the fuel bed.

The reaction between the carbon and oxygen of the air has been considered, and it has been pointed out that for each pound of carbon burning to carbon dioxide, 14 540 B. t. u. was given out; also, that if this carbon dioxide were decomposed into the monoxide, its formation would take up 10 100 of the B. t. u. originally given out. Consequently, 4440 B. t. u. would be left in the form of sensible heat, part of which might be utilized in the decomposition of water into oxygen and hydrogen. The hydrogen liberated through such a reaction would pass through the fuel bed partially intact, thus adding a combustible constituent to the gas, while the oxygen would unite with the carbon to form either the dioxide or the monoxide, depending upon the temperature at which the reaction occurred. It is considered, ordinarily, that the following reactions occur within the producer when the vapor of water is introduced with the air, viz:

$$2\,H_2O+C=2H_2+CO_2 \ldots\ldots\ldots\ldots\ldots\ldots (3)$$

and

$$H_2O+C=H_2+CO \ldots\ldots\ldots\ldots\ldots\ldots (4)$$

Reaction 3 predominates in the region of low temperatures between 1000° and 1600° F. Above 1600° F., reaction 4 predominates. At a temperature close to 1000° F., there will scarcely be a trace of reaction 4. If it is considered that the fuel bed is of a uniform temperature, in the neighborhood of 1000° F., so that reaction 3 holds, the following relation exists between the constituents entering into the reaction.

36 lb. of water + 12 lb. of carbon produce 4 lb. of hydrogen and 44 lb. of carbon dioxide, i. e.,

$$2\,H_2O + C = 2H_2 + CO_2$$
$$36 + 12 = 4 + 44$$

or

$$3 + 1 = \tfrac{1}{3} + 3\tfrac{2}{3}$$

and 1 lb. of carbon burning to CO_2 liberates 14 540 B. t. u.,
1 lb. of hydrogen burning to H_2O liberates 62 000 B. t. u.

Since $\frac{1}{3}$ lb. of hydrogen is formed through this reaction, $\frac{62\,000}{3} =$ 20 660 B. t. u. disappears or is absorbed. For each pound of carbon entering into the reaction, $20\,660 - 14\,540 = 6120$ B. t. u. disappears. Evidently, the above heat deficit must be supplied by additional carbon in the fuel bed burning to CO_2 through the presence of oxygen supplied in the air and according to reaction (1) $C + O_2 = CO_2$, since for the assumed temperature of the fuel bed, it will not be possible for more than a trace of the carbon monoxide to form.[*]

From equation (1), 14 540 B. t. u. is liberated per pound of carbon, therefore $\frac{6120}{14540} = .42$ lb. additional carbon, burned to CO_2, is necessary to supply heat for the completion of reaction 3. Consequently, 1.42 lb. of carbon are necessary for the formation of $\frac{1}{3}$ lb. of hydrogen, all of the heat of the carbon being utilized in the production of hydrogen, which is the only combustible constituent of the gas, the other constituents being CO_2 and N_2.

In considering reaction 4, which predominates at the higher temperature, it will be well to refer again to the experiments of Harries.

---

[*] Haber's "Thermodynamics of Technical Gas Reactions", page 138.

This has been illustrated very clearly by the experiments of Harries. Harries passed water vapor over incandescent carbon in a tube at various temperatures and found the relation between the CO, CO_2, H_2O and H_2 in the resulting gas. At a temperature of 1240° F., the following results were obtained:

H_2 per cent = 8.41; CO_2 per cent = 3.84; CO percent = 0.63; H_2O per cent = 87.12

The extent of this reaction, doubtless, depends not only upon the temperature, but also upon the time of contact of H_2O and CO_2 with the carbon, and while the latter variable has evidently been neglected in the experiments, the results are sufficiently decisive to justify the above assumption that at the low temperature, the percentages of CO formed will be a negligible quantity.

At a temperature of 2050° F., the following relation was found between the constituents of the gas leaving the tube,

$$\begin{aligned} H_2 \text{ per cent} &= 50.730 \\ H_2O \text{ per cent} &= 0.303 \\ CO_2 \text{ per cent} &= 00.600 \\ CO \text{ per cent} &= \underline{48.340} \\ &\ 99.973 \end{aligned}$$

This clearly illustrates the predominance of reaction 4.

If it is assumed again that the temperature throughout the fuel bed is uniform and is such that only reaction 4 occurs, we have

18 lb. water + 12 lb. carbon producing 2 lb. hydrogen and 28 lb. CO

or

$$\begin{aligned} H_2O + C &= H_2 + CO \\ 18 + 12 &= 2 + 28 \end{aligned}$$

or

$$1\frac{1}{2} + 1 = \frac{1}{6} + 2\frac{1}{3}$$

1 lb. of carbon burning to CO liberates 4440 B. t. u.,

and

1 lb. of hydrogen burning to H_2O liberates 62000 B. t. u.

Consequently, the formation of $\frac{1}{6}$ lb. of hydrogen absorbs $\frac{62000}{6}$ = 10330 B. t. u. A deficit of 10330 — 4440 = 5890 B. t. u. results. This may now be made up by carbon burning to CO through the oxygen supplied in the air according to reactions 1 and 2. For each pound of carbon burning to CO 4440 B. t. u. is liberated. Consequently, in order that reaction 4 be completed, $\frac{5890}{4440}$ = 1.33 lb. of carbon must burn to CO. The total heat from the burning of 2.33 lb. of carbon to CO results in the production of $\frac{1}{6}$ lb. of hydrogen. $1\frac{1}{2}$ lb. of water are theoretically necessary, or .64 lb. of water per lb. of carbon.

The gas resulting from the above reactions consists of CO, H_2, and N_2. The percentage by volume may be obtained in the following manner:

Total weight of $H_2 = \frac{1}{6}$ lb.
Total weight of $CO = 2.33 \text{ x } 2\frac{1}{3} = 5.44$

Total volume of H_2 at 62° F. and 30 in. Hg. = $\frac{1}{6}$ ÷ 0.0053 = 31.5 cu. ft.
Total volume of CO at 62° F. and 30 in. Hg. = 5.44 ÷ 0.0736 = 74.00 cu. ft.
Total volume of O_2 from air $1\frac{1}{3}$ x 1.33 ÷ .0842 = 21.1 cu. ft.
Total volume of air 21.1 ÷ 0.21 = 100.3 cu. ft.
Total volume of N =79.2 cu. ft.
Total volume of gas leaving producer 31.5 + 74 + 79.2 = 184.7 cu. ft.
Total volume of gas per lb. of carbon 184.7 ÷ 2.33 = 79.3 cu. ft.

Theoretical Gas Analysis	High Heating Value B. t. u. per cu. ft.
H_2 per cent = 17.0	55.8
CO per cent = 40.0	127.8
N_2 per cent = 43.0	000.0
total	183.6

4. *Conditions in the Actual Producer.*—In the actual producer, reactions 3 and 4 are doubtless taking place continuously in different parts of the fuel bed. It would hardly be possible to operate a producer at so low a temperature as to produce the results obtained according to reaction 3. It would be impossible to operate at such a temperature as to prevent reaction 3 occurring.

The conditions that probably maintain are as follows. The moisture laden air, comparatively cool, passes into the fuel bed; on entering, it cools down the first layer of fuel, or the combustion zone, as it is usually called, to such an extent that reactions 1 and 3 probably result, part of the oxygen supplied by the air uniting with the carbon to form CO_2, while part of the moisture decomposes and forms H_2 and CO_2.

The carbon dioxide, formed according to reactions 1 and 3, passes up into the hotter portion of the fuel bed and takes up other atoms of C to form CO, according to reaction 2. The moisture which is not decomposed and the O_2 which is not combined in the combustion zone pass into the hotter portion of the fuel bed known as the decomposition zone, or dissociation zone, where reactions 1 and 2 probably take place in immediate succession, in the case of the oxygen, while part of the moisture is combined according to 4, producing H_2 and CO. The gases leaving this portion of the fuel bed are therefore composed of H_2, CO, CO_2 and small quantities of O_2, and vapor of water, the last two constituents having either passed through the fuel bed intact or having resulted from dissociation. Theoretically, at the higher temperatures there should be only a trace of CO_2; actually, this quantity will vary from 2 to 17 per cent of the volume of the gas, depending upon a number of conditions. The gases passing from the dissociation zone or layer of high temperature enter into the distillation zone, which is at a lower temperature, and which is so named for the reason that in this zone the volatile matter is distilled from the fresh fuel by the hot fuel bed beneath. This volatile matter for anthracite coal consists of small quantities of H_2, H_2O, CH_4, C_2H_4 and condensible hydrocarbons in the form of tar, etc. The CH_4 and C_2H_4 in the producer gas made from anthracite are inconsiderable; the former will not represent more than 2 per cent of the volume of the gas, while the C_2H_4 will probably not exceed 0.1 per cent by volume. The condensible hydrocarbons are also inconsiderable.

In producer gas made from bituminous coal, these distillation products may represent 40 per cent of the heating value of the gas, and are not only desirable but necessary constituents, when the gases are to be used for reverberatory and other metallurgical furnaces where high temperatures are desirable.

The following reaction, which may occur between the constituents of producer gas, is reversible and depends upon the temperature.

$$CO + H_2O = CO_2 + H_2 \ldots\ldots\ldots\ldots\ldots\ldots (5)$$

From the results of Hahn's[1] investigations, it was shown that at a temperature of 1520° F., H_2 and CO became equally strong reducing agents. At lower temperatures, the carbon monoxide is the stronger. This means that if the gases on leaving the dissociation zone enter a zone at a temperature lower than 1520°, there will be a tendency for the CO to react on the water vapor present and form CO_2 and H_2. At higher temperatures, there will be a tendency toward the formation of CO and water. The velocity of the reaction and the extent to which each takes place depend upon the temperature and the depth of the fuel bed. High temperatures and deep fuel beds tend to produce a gas low in CO_2 and H_2. Low temperatures or shallow fuel beds produce a gas low in CO, and high in CO_2 and H_2.

It should be understood that the above reactions depend very largely upon the presence of the fuel which acts as a catalyst. Allner[2] has shown that if the above reaction in the presence of a catalyst is established, or if equilibrium maintains at a temperature above 2200° F., and if the gases are cooled without the presence of the catalyst, the reactions are "frozen", that is, the removal of the catalyst does not permit the readjustment of equilibrium due to the lower temperature that would otherwise have occurred had it not been removed. If, on the other hand, the catalyst is not removed and the gases are cooled, the reaction will continue until a temperature of about 1400°F. is reached. This latter is the condition that maintains in the producer as the gases flow from the dissociation zone to the distillation zone. The extent of this reaction depends upon the velocity of the gases through the distillation zone, the depth of this zone, and upon the temperatures within the zone. After the gases have left the zone, i. e., are out of the presence of the catalyst, the reaction is "frozen", and no further change in the composition of the gas occurs.

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[1] Haber: Thermodynamics of Technical Gas Reactions, page 145.
[2] Haber: Thermodynamics of Technical Gas Reaction, page 309.

5. *The Carbon Monoxide Producer.*—The simple carbon monoxide producer has been considered in the discussion of the theory of the producer. In contradistinction to this, there is a commercial producer known as the carbon monoxide producer, which is being used to some extent. The differentiation of this producer from other commercial producers lies in the substitution of CO_2, taken from some outside source, for the vapor of water, for the purpose of conserving part of the 30 per cent heat loss inherent in the simple carbon monoxide producer. The producer is used principally for the driving of gas engines. The supply of CO_2 is obtained by piping the exhaust from the engine to the ashpit of the producer.

From reactions 1 and 2, and when one pound of carbon burns to CO, it is known that 4440 B. t. u. is given out, which may be utilized in the dissociation of CO_2. 14 540 B. t. u. is liberated per pound of carbon entering into the reaction, and since in the production of CO, 4440 B. t. u. is liberated, evidently 10 100 B. t. u. remains in the CO. Therefore, in the reduction of CO_2, 10 100 B. t. u. is absorbed per pound of carbon entering into the reaction, while $\frac{10100}{3\frac{2}{3}} = 3966$ B. t. u. is absorbed per pound of CO_2. The

B. t. u. free for the carrying out of this reaction per pound of carbon is 4440. The number of pounds of CO_2 necessary for the utilization of this heat is $\frac{4440}{3966} = 1.11$ lb. Since each pound of carbon in the producer appears principally as CO_2 in the exhaust gas of the engine, there will be $3\frac{2}{3}$ lb. of CO_2 produced in the exhaust per lb. of carbon in the producer which is much more than is necessary for the above reaction.

On investigation, this system would seem to offer a number of advantages over the system in which the producer uses steam for the production of H_2. The gases leaving the producer consist principally of CO and N_2, together with small quantities of H_2, CH_4, and C_2H_4, distilled off from the green fuel. The gas, therefore, will be likely to be more uniform in quality than H_2 enriched gas, and will, consequently, be less likely to cause premature ignition in the engine cylinder. This will be due to the large quantity of nitrogen present in the gas and to the small quantity of hydrogen. It will also be possible to use a much higher compression in the engine cylinder, and this will tend to offset any loss in efficiency in the producer that may be caused by the sensible heat lost in the nitrogen. Since part of the engine exhaust is delivered to the producer, the sensible heat in the exhaust is utilized in addition to the heat contained in the unburned products resulting from incomplete combustion in the engine cylinder. These latter may represent anywhere from 5 to 20 per cent of the heating value of the original gas. The disadvantage of the process doubtless lies in the regulation of the amount of CO_2 delivered to the producer.

II. PURPOSE OF THE TESTS AND DISCUSSION OF THE METHODS OF EXPERIMENTATION

6. *Object of the Tests.*—In the field of small isolated power plants, the suction gas producer using anthracite coal of the finer grades has for the past few years been fighting for place. As in the case of almost all new apparatus, its ultimate success or failure has been retarded largely on account of the lack of impartial data on the efficiency, cost of operation, reliability, etc., together with the natural prejudices against change, and the difficulty in

securing capable operators. The tests herein described were made for the purpose of obtaining impartial data on the efficiency, reliability, and operation of suction gas producers of small size, using anthracite coal as fuel. They were conducted on the producer in the Mechanical Engineering Laboratory, of the University of Illinois. Incident to the main object of the tests was the development of a method of studying and testing the producer, the standardization of forms for the presentation of the results of the tests, and probably the most important of all, the derivation of formulas for the necessary computations. Twenty-five tests were made and four grades of fuel used. The results of all tests are included in Table 5, and the forms and formulas are given in the Appendix.

It is hoped that the results of the tests themselves will be of value to manufacturers and to users of small power; the method and forms, a partial discussion of which has appeared in a previous article[1], may be of value to builders of gas producers and to engineers engaged in development and testing.

7. *The Producer.*—The producer under consideration was installed by the Otto Gas Engine Works of Philadelphia, and is known as their No. 3 producer. In the specifications, it was stated that the producer had a maximum capacity of supplying gas for 60 horse-power, and that this was equivalent to a maximum production of 8100 cu. ft. of gas per hour. It was also stated that the producer was designed for intermittent operation and that the length of runs should not be greater than 12 hours.

TABLE 1

DIMENSIONS AND PROPORTIONS

1.	Dimensions of grate, ft.	1.25 x 1.33
2.	Grate area, sq. ft	1.666
3.	Mean diameter of fuel bed, ft	1.545
4.	Depth of fuel bed, ft	2.21
5.	Area of fuel bed, sq. ft	1.877
6.	Height of discharge pipe above grate, ft	2.875
7.	Approximate width of air spaces in grate, in,	0.5
8.	Area of air space, sq. ft,	0.722
9.	Proportion of air space to whole grate area, per cent	43.3
10.	Area of discharge pipe, sq. ft	.165
11.	Outside diameter of shell, ft	2.833
12.	Length of shell from base to top of magazine, ft	7.125
13.	Ratio of minimum draft area to grate area, 1 to	48.8

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[1] Journal A. S. M. E., Dec. 1909.

The producer, the principal dimensions of which are given in Table 1, was of the contained vaporizer type and was provided with a plain bar grate.

The plant, as installed by the Otto Gas Engine Works, consisted of the producer, a wet scrubber, a gas receiver and a 23 horsepower producer gas engine known as their No. 7 engine. A section through the producer is shown in Fig. 2. A diagrammatic sketch of the plant as modified for testing is shown in Fig. 3.

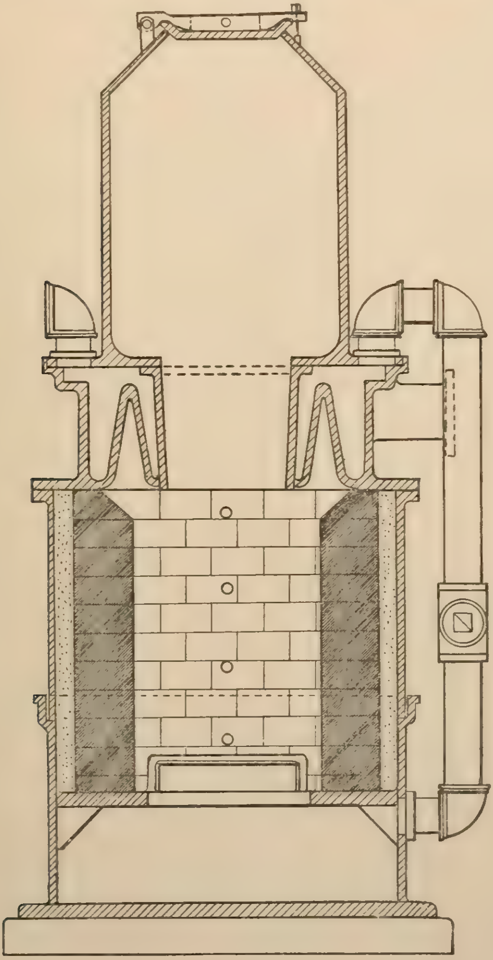

FIG. 2

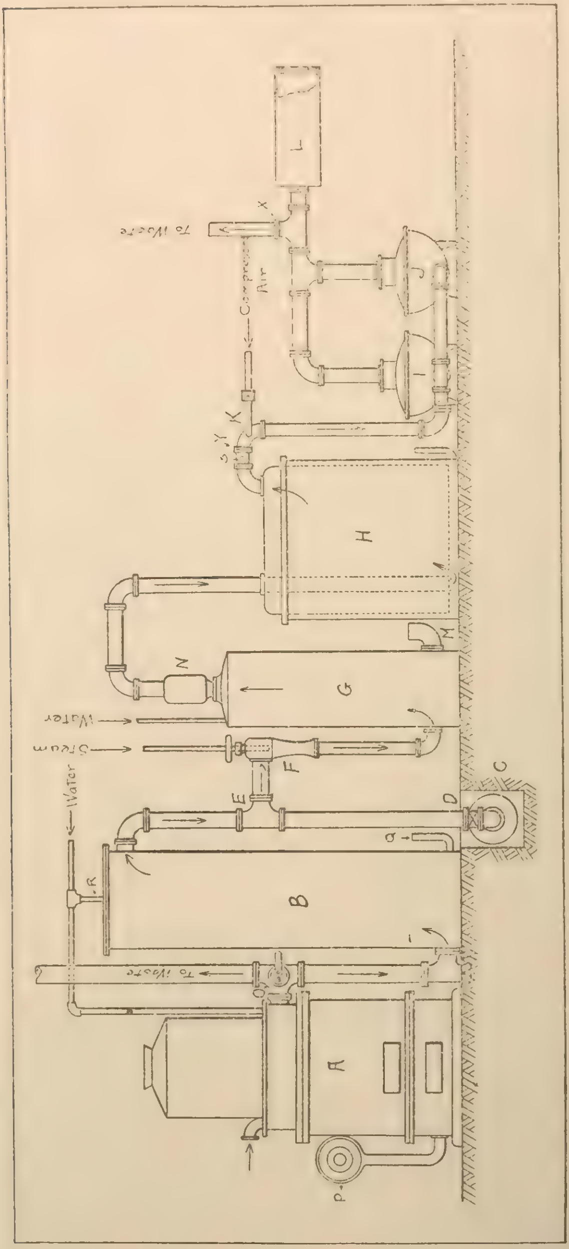
A
B
C
D
E
F
G
H
I
J
K
L
M
N
P
Q
R
S
X
Y
Water
Water
Steam
To Waste
To Waste
Compressed Air

FIG. 3

In order to relieve the producer test of an engine running in conjunction with it and to make the former independent of the latter, it was decided to blank off the engine from the gas main and produce the necessary suction by means of a steam ejector. This reduced the labor of operating to minimum, made regulation positive, under immediate control, and produced conditions tending to insure greater accuracy and satisfaction.

It has been urged that producer tests can not be run like boiler tests, independently of the engine, principally on account of the prejudice existing among prospective buyers of producer installations. This objection would doubtless hold in the case of acceptance tests, but even so, it would be desirable in all such cases to run the engine tests in conjunction with the producer tests, and under the normal conditions of everyday operation.

However, for the purpose of studying the producer when operating under different conditions, or for studying the action of different fuels in the producer, or for obtaining data on the efficiency and composition of the producer gas generated, the present method has decided and obvious advantages.

8. *The Plant.*—Referring to Fig. 3, the producer A is shown, provided with a fan blower P for starting, a two-way cock O, a waste pipe leading to the roof and a water seal in the connection between the producer and the first wet scrubber B. The wet scrubber B was filled with coke and provided with an overflow at Q. The scrubber water entered at R. The Schutte-Koerting steam ejector used for producing the draft, was located at F and provided with the steam connection as indicated. The ejector has a capacity of 12 000 cu. ft. of gas per hour. The steam used by the ejector is condensed in the second wet scrubber G and the suspended moisture is removed by the separator N and the dryer H. The latter is simply a gas bell filled with straw. Its use was made necessary by the Westinghouse meters located at I and J. These meters are of the "wet" type, so that the suspended moisture in the gases, before the use of the dryer, tended to collect in the meter and raise the level of the sealing fluid. The gauge box L was blanked off from the pipe line during the producer tests and was used only for the calibration of the meters. This calibration was effected by the use of air introduced at the compressed air connection indicated.

9. *Method of Conducting the Tests.*—In conducting the tests,

the items to be considered first were the weight of coal, the heating value of the coal, the volume of gas, and the heating value of the gas. The measurement of these quantities will be considered in the order named.

(1). *Determining the Weight of Coal Fired.*—The correct determination of the weight of coal fired in testing producers of the intermittent type, where tests of short duration are necessary, is a problem of no small importance, owing to the difficulties in obtaining the weight of coal in the producer at the start and at the close of the test. The method of starting that suggests itself as being the most readily carried out, especially by the engineer, is to clean the ash and clinker out and work the fuel bed, by means of the poking bar, into a uniform condition; then, in closing, to duplicate these conditions as nearly as possible.

In any actual case, it is always possible to compute from a number of actual trials the average weight of coal required to fill the producer to a given level. If this mean value is taken as the true value or true weight of coal required to fill the producer, then the maximum variation from the mean or true value in the case of any one trial will indicate the probable maximum error that will be made in filling the producer. Evidently this same maximum error may be made in bringing the fuel bed to the starting condition, irrespective of the weight of coal required to bring the fuel bed to this condition. An example will illustrate this better. Suppose that for a number of actual trials, the average weight of coal required to fill the producer to a given level is 600 lb.,—suppose the maximum variation from the mean is 15. The maximum error is 2½ per cent, based on 600 lb. Suppose, further, that in a given test, 200 lb. of coal were burned. Evidently in bringing the fuel bed to the starting condition and in filling, a maximum error of 15 lb. may be made; i. e., in filling, it is possible to make an error in this particular test of 7½ per cent.

In order to determine the error approximately in estimating the weight of coal during the present tests, the producer was filled four separate times, and the weight of coal required noted in each case. The average of the four weights was taken as the mean weight or true weight of coal required to fill the producer. The results are given in Table 2.

It will be seen from this table that the maximum variation from the mean is 8.75 lb. or 1.7 per cent. It will be seen from Table 5, giving the results of the tests, that the smallest weight of coal fired on any test was for test 26, and that this weight was 146 lb. In bringing the fuel bed to starting conditions and in filling the producer at the close of the test, it would, therefore, have been possible to make an error of 8.75 lb., or, in this case, an error of $\frac{8.75 \text{ x } 100}{146} = 6$ per cent. This is, of course, the purely mechanical error in filling, and does not take into consideration the difference that may maintain in the condition of the fuel bed in starting and stopping. There is always present in the fuel bed a larger per cent of ash in stopping than in starting, and if this is in the form of clinker, a large error may result. If it is in the form of a powder, the error will probably not be so great, as the ash will tend to pack into the interstices around the coal; and while the fuel bed may contain a larger per cent of ash in stopping than in starting, the volume of the fuel bed, and the weight of carbon present may remain practically the same. The error, therefore, in the estimation of the weight of coal due to the presence of the ash will not be large. The error in determining the weight of ash is unimportant, as this may be determined from the analysis of the coal and the total weight of coal fired, with a greater degree of accuracy than could possibly be determined from the weight of the ash and refuse taken from the producer.

TABLE 2

Weight of Green Coal Required to Fill the Producer

Trial No.	Weight pounds	Variation from Average Weight	Variation per cent
1	669.25	8.75	1.70
2	676.25	1.75	.26
3	683.25	+5.25	.77
4	683.25	+5.25	.77
Total	2 712.00		
Aver.	678.00		

The above method of starting and stopping the test has been used throughout the present work. There is another and much more accurate method of procedure, which is, however, both tedious and expensive. Fill the producer with a weighed amount of wood and coal and start the fire. During the period of starting, measure the volume of gas discharged, and at the same time take a continuous sample for analysis. Also weigh the ash and refuse accumulating during this period, and take a sample for analysis.

Reduce the total volume of gas to standard gas, i. e., gas at 62° F. and 30 in. Hg., and calculate from the analysis the volume of CO_2, CO, H, CH_4 and C_2H_4. From the specific weights of the constituents of producer gas as given in the Appendix page 84, item 121, calculate the total weight of each of these constituents. Let a, b, c, d, e, be the weights of the respective constituents. Then the total weight of carbon that has been taken from the fuel bed during the starting period will be,

$$W = \tfrac{3}{11} a + \tfrac{3}{7} b + \tfrac{3}{4} d + \tfrac{6}{7} e + w_1$$

where w_1 = the total weight of carbon as determined from the analysis and the weight of the ash and refuse.

The total weight of hydrogen that has been taken from the fuel in the producer and that appears in the gas may be determined by the formula

$$W_2 = c + \tfrac{1}{4} d + \tfrac{1}{7} e$$

This neglects the hydrogen that may be formed by the decomposition of water, and will, therefore, introduce an error, the amount of which will depend upon the length of time between the lighting of the fire and the starting of the test. With small producers, this will be very small. In large producers, tests are usually of such length and the weight of coal fired of so large an amount, as to make the error in the estimation of the coal in starting and stopping a negligible amount, consequently, such a method as the one under discussion would not be necessary.

In closing the test, the coal should be burned low in the producer, and immediately after closing, the fuel bed should be drawn and weighed, the incandescent coals quenched and then sampled, and an analysis made from the sample.

The total weight of equivalent coal fired during the test may be obtained from the following formula,

$$W_c = W_s + W_t - \frac{(W + W_f)\ 14\,560 + W_2 \text{ x } 62\,000}{H}$$

where

W_c = total weight of equivalent coal fired during the test.
W_s = total weight of equivalent coal in producer at the start.
W_t = total weight of coal fired during test.
W = total weight of carbon appearing in the gas before starting.
W_2 = total weight of hydrogen appearing in the gas before starting.
W_f = total weight of carbon within the fuel bed at the close of test.
H = The heating value of the coal per pound.

(2) *Heating Value and Analysis of the Coal.*—The heating value and the analysis of the coal were determined by the chemist, and a discussion of the chemical method is unnecessary here. The chemical work is probably accurate within one per cent. The greatest error in the analysis or in the heating value is likely to be made in the sampling of the coal. The weight of the coal sample should be not less than 10 per cent of the total weight of coal fired, and the samples taken in the present tests represented from 10 to 15 per cent of the weight of coal fired. These samples were mixed and quartered until a sample sufficient to fill about eight quart jars was obtained; this sample was then reduced by grinding until it would pass a $\frac{1}{4}$ in. screen. This was again mixed and quartered, and a sample taken sufficient to fill a pint jar. This sample was sent to the chemist for analysis.

The coal used in the tests was stored indoors and was practically air-dry as fired. Proximate analyses and heating values were obtained from the coal used in every test. Ultimate analyses were obtained for at least three samples from each grade of coal used. From these ultimate analyses and the proximate analyses from each test, the ultimate analysis for each test was determined by a method of approximation.

(3) *Measuring the Volume of the Gas.*—The volume of the gas generated was measured by Westinghouse meters. These were of the wet type, so that the gas should contain no suspended moisture. This moisture was removed by the dryer shown in Fig. 3. In order to insure accuracy in the measurement by the meters, a gauge box *L*, Fig. 3, was connected to the gas main so that the meter or meters could be calibrated from time to time. This calibration was effected by the use of compressed air. Blind flanges were

placed at X and Y, and the air was admitted at the connection shown. This air, expanding, passed to the meter and then to the gauge box which contained a thin plate orifice. The pressure and temperature of the air were taken at the meter and also at the gauge box. From the pressure and temperature in the gauge box the volume of air passing the orifice was computed from the work of R. J. Durley.[1] From this volume, and the pressure and temperature taken at the meter, the volume passing the meter was computed.

The calibration was made at different capacities and a calibration curve for the meter plotted. This latter was practically a straight line.

In obtaining the volume of gas generated during a test, the volume was taken from the calibration curve of the meter and then reduced to standard gas by the formula given on page 86, of the Appendix, item 125. The volume of gas as obtained from the meter, as just explained, was checked by calculating the volume of gas generated from the weight and analysis of fuel and the analysis of the gas. The formula for this computation is given in the Appendix, page 86, item 126.

The volumes as obtained by computation ordinarily checked within 5 per cent of the volumes determined by the meters. Where the sampling of coal and ash is carefully carried out, this method of computing the volumes from the analysis is reliable for producers using hard coal. It is based on the fact that the weight of carbon contained in the coal must equal the weight of carbon contained in the gas plus the weight contained in the ash and refuse, plus the weight lost in tar and the weight lost in the gas absorbed by the scrubber water. The carbon lost from the hard coal producers in the form of tar and condensible hydrocarbons is very small, doubtless less than one per cent. The weight lost by the absorption of CO_2 and CO in the scrubber water is also very small, so that the carbon in the coal should be accounted for within 5 per cent at the most. The method may even be used for the determination of the volume of gas generated by bituminous coal producers. In the case of these producers, 10 per cent of the carbon may be lost in the tar and other heavy hydrocarbons which condense and are deposited in the scrubbers and gas mains.

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[1] Trans. American Society Mechanical Engineers, Dec. 1905.

The percentage of carbon so lost may be found by taking a continuous sample of the gas as it leaves the producer, and before it is cooled, and drawing it through some form of condenser in which the condensible matter will be thrown down while the gas passes to a small meter for measurement. From the weight and analysis of the condensed matter and the volume of the gas sample, the per cent of carbon lost in this way may be obtained. The determination is tedious and requires considerable outlay, but it may often be used to advantage in the testing of large producers where mechanical means of measuring the volume of the gas generated are impossible. The value of the volume of gas computed from analysis as a check on the accuracy of the test has been discussed on page 39.

(4) *Sampling the Gas*[1].—The correct sampling of the gas for analysis and for the determination of the heating value is imperative. In the present tests, the form of sampling device is illustrated in Fig. 4. This device takes the sample from three separate points in the main and practically the same volume of gas from each point. In mains of larger diameter, more nipples should be used. A good rule is to use one sampling nipple for each inch of diameter of main.

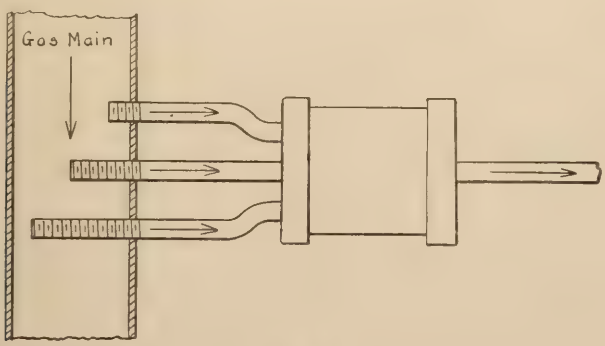

FIG. 4

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[1] Credit is due Mr. O. A. Carnahan for his painstaking and conscientious work in the analysis of the gases, for all the tests, and his ingenuity and suggestions along this particular line. Credit is also due Mr. J. P. Clayton for his assistance in computing, and in operating the calorimeter on a large number of tests.

The above device was inserted beyond the second scrubber and at S, Fig. 3. In this position, the gas is under pressure so that it is necessary only to connect the aspirator bottle used for collecting the sample of gas to the nipple leaving the sampling device.

Where the gas is under a pressure lower than atmospheric, it is necessary to connect an aspirator of the ordinary laboratory type to the sampling device and draw the gas from the main by the aspirator and force it into a vessel provided with a water seal. Samples of the gas may be taken from this vessel both for analysis and for the calorimeter. Fig. 5 illustrates the arrangement. The principal objection to its use is that small quantities of gas, principally CO_2, are absorbed by the water in the aspirator.

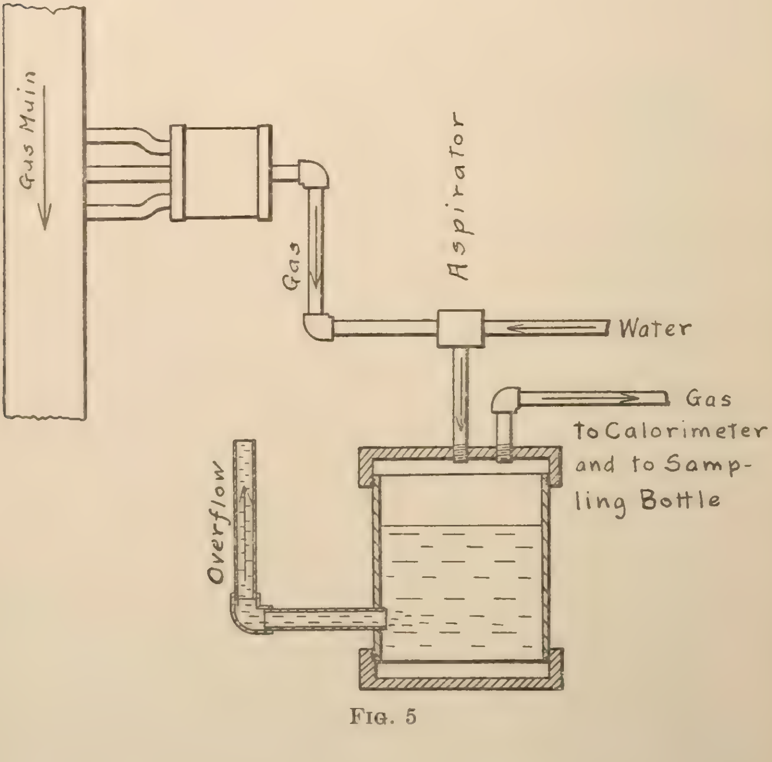


FIG. 5

The sample of gas taken from the sampling device for analysis was collected for the present tests in ordinary aspirator bottles and over water previously saturated with producer gas. These bottles held about 5 litres. 600 cc. of the sample was transferred to a special glass sampling tube containing mercury. This sample was then sent to the chemist for analysis. The sample of gas for the Junker calorimeter was also taken from the above sampling device by connecting the burner of the calorimeter through rubber tubing to the nipple. When operating the calorimeter,

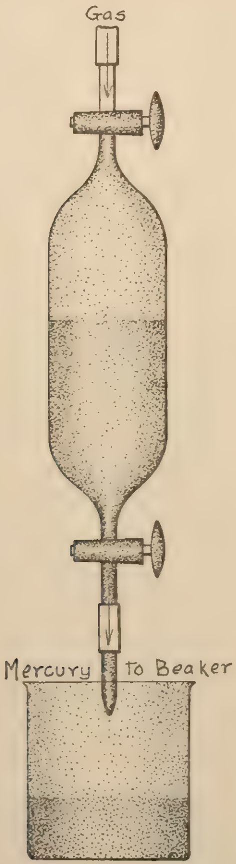


FIG. 6

samples were taken continuously, one after the other.

In order to obtain correct samples of gas, they should be drawn from the main continuously and at a uniform rate of flow. With the aspirator bottles above mentioned, the condition of uniform flow was practically impossible at the time of sampling.

Later, the gases were collected in glass sampling tubes containing mercury. Each tube held about 600 cc. The tubes were connected directly to the nipple leading from the sampler, and the mercury was drawn off through a glass cock located at the bottom of the tube. The mercury passed through the cock and dropped from a glass tube, the opening in the end of which had been made of such size as to allow the mercury to run from the tube in about two hours' time. Fig. 6 illustrates this arrangement. The apparatus has the advantage of collecting the sample over mercury, which eliminates small errors that alwaysresult from the absorption or liberation of gases by the water, and in addition permits the observer to see at a glance whether or not stoppage of the flow of mercury has occurred.

(5) *Determining the Heat Value.*—The heating value of the gas has been determined by the Junker calorimeter and by computation from the analysis of the gas. The latter value is assumed to be more nearly the true value, and has been used in the computations. It is assumed to be more nearly the true value for the reason that the gas samples from which the analyses were made were continuous samples taken over the entire period of the test, while samples for the Junker calorimeter were more or less intermittent, due in the case of some of the earlier tests on the Philadelphia and Reading coal, to poor gas, which would not burn at times, and again, to the necessity of using the operator on other work. There are, also, certain errors such as radiation and conduction and possibly errors in the meter which enter into the calorimeter determinations and for which allowance cannot readily be made. Again, the thermometers used in measuring the temperature of the entering and leaving water read only to tenths of a degree so that an error of 0.5 of one per cent could easily be made in estimating to hundredths, as the rise in temperature of the water was frequently not greater than two or three degrees.

It was found by actual trial that radiation and conduction could easily affect the readings to the extent of two per cent in making determinations upon producer gas. The temperature of the water used in the calorimeter in the case of all tests was below the room temperature, so that the heat was conducted into the calorimeter. This tended to make all determinations about two per cent high. Upon calibrating the gas meter used with the calorimeter, it was found to read a fraction over two per cent low. The error in the meter was, therefore, considered to offset the error due to conduction and radiation.

On account of the above causes, it is believed that the heating value of gases whose principal constituents are simple gases, such as CO and H_2, may be determined with a greater degree of accuracy by computation from the analysis rather than by the calorimeter, especially where the latter is operated under conditions existing on a test and possibly by one unfamiliar with the errors likely to be made in using the instrument.

In the case of all the present tests on which the calorimeter was run continuously, the heating value as determined by the calorimeter checked the heating value as determined from analysis within less than three per cent, and on a number of these within one per cent. This seems to indicate that either method of obtaining the heating value is accurate if proper care is used.

(6) *Error Due to Vapor Pressure of Water.*—An error that caused considerable discrepancy between the heating value as given by the calorimeter and that as given by the analysis, and which resulted in about ten days of investigation upon the accuracy of the Junker calorimeter, was the neglect of the effect of the vapor pressure of water contained in the gas passing through the meter. This source of error had been recognized in the deriving of the formula for the calorimeter that appears in the Appendix, page 83, item 120, and had been allowed for, but in temporarily working up the heating value on some of the tests, this formula was not used, and this error was lost sight of. The result was an error of from two to four per cent, which could not be accounted for after allowing for radiation and other errors in the use of the apparatus. As we have since found that this is an error that is not usually recognized or allowed for by engineers, it will not be out of place to discuss and indicate its magnitude.

The gas entering the Junker meter is saturated with the vapor of water. Consequently this vapor is under the pressure due to the temperature of the gas. According to the laws of Dalton, if a mixture of gases is inclosed within a vessel, the volume of each gas occupies the entire volume of the vessel and each gas is under its own pressure. The sum of the pressures of the constituents of the mixture is equal to the total pressure within the vessel.

The total absolute pressure within the meter is the barometer pressure plus the pressure in inches of water indicated by the U tube attached to the pressure regulating device that accompanies the meter. The temperature of the gas within the meter is obtained from the thermometer. Assume that the total pressure in the meter is 30 inches of mercury absolute, and that the temperature of the gas is 70° F. We find from the steam table that the vapor pressure of the water is .73 in., the total pressure of the mixture is 30 in., consequently, the pressure of the dry gas is 30.0–0.73=29.27 inches. If the temperature in the meter is 100° F., the pressure of the vapor is 1.94 in., and the pressure of the dry gas is 30.0–1.94=28.06 in. of mercury.

From these examples, it will be seen that an error of about $\frac{.73}{30} \times 100 = 2.4$ per cent will be made at 70°F. by neglecting the effect of the vapor pressure in computing the standard volume of gas used by the calorimeter and that an error of about $\frac{1.94}{30} \times 100 =$ 6.5 per cent at 100° F. will be made. The same error will be made in computing the total volume of gas discharged from the producer, as indicated by the gas meters.

The effect of the above error is to give a low value for the heat of combustion of the gas, and to give a high value for the total volume of gas generated by the producer. If, therefore, no correction is made, the total heat contained in the gas generated by the producer, since it is the product of the two quantities, will not be in error, and the efficiency of the producer will consequently not be in error. If, however, the total volume of gas generated has been corrected for vapor pressure, and the volume of gas used by the calorimeter has not been corrected, which is frequently the case, the efficiency of the producer will be low by the amount of this error.

(7) *Analyzing the Gases.*—The analysis of the gas was made in the Hempel gas analysis apparatus and over water. The burette used was water-jacketed to prevent changes in temperature

affecting the results, while the water used in the burette was saturated with gas.

In some of the later tests, the gases were collected in gas sampling tubes holding about 600 cc. and were collected over mercury. The analysis was also made over mercury. It is believed that this was at least, a safe precaution, as the water in the aspirator bottles previously used for sampling, and in the burette, even if saturated with the gas beforehand, might either absorb or give off small quantities of the gas. There are also other advantages in the use of the "accurate" method which need not be discussed here.

(8) *Measuring the Weight of Air Used.*—As the weight of air is not used in calculations of the first importance, it was not considered necessary to measure this quantity, as it could be computed from the weight of nitrogen appearing in the gas with an accuracy well within 5 per cent. This has been done for all tests and the result appears under item 97 of Table 5.

(9) *Measuring the Temperatures.*—All temperatures with the exception of the temperatures in the fuel bed and the temperature of the gases leaving the producer, were taken with mercury thermometers, all of which had previously been calibrated and calibration curves plotted. All mercury thermometers were used with their bulbs in direct contact with the medium whose temperature was to be measured. In calibrating, the Reichsanstalt standards of the department of Physics were used. In making the computations, the corrections were taken from the calibration curves.

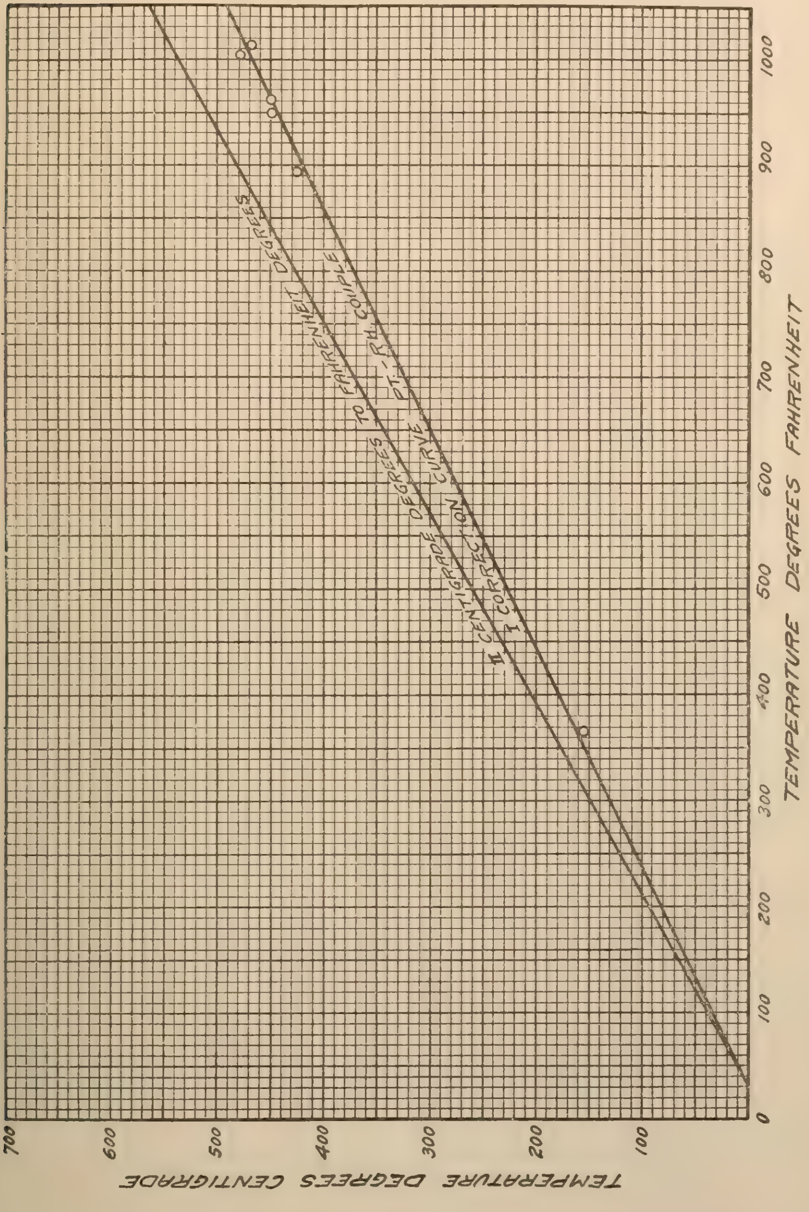
(10) *Temperatures of Gases Leaving Producer.*—The measurement of the temperature of the gases leaving the producer was effected by the use of a platinum-rhodium thermocouple and a Siemens & Halske millivoltmeter calibrated to read direct in degrees Centigrade. This couple and voltmeter were compared with a Reichsanstalt standard, reading in degrees Centigrade and to 500°. After making the proper stem correction for the standard, the readings of the thermocouple practically agreed with the readings of the standard. For higher temperature, the thermocouple readings were taken at the melting point of zinc, 419° C., the melting point of silver, 961° C., and the melting point of copper, 1084 ° C., and were found to be practically correct. In using the platinum-rhodium couples, considerable care had

to be taken in order not to have the couple contaminated with gases, as such contamination would destroy the calibration and ultimately make the couple worthless. As a couple of this type five feet in length costs about $50.00, it is evidently necessary to use some care in this respect. In order to protect the present couple, the hot junction was placed in a quartz glass tube about ⅛-in. inside and ¼-in. outside diameter. This tube was then inserted in a brass thermometer cup which was screwed into the gas main leaving the producer at *O*, Fig. 3. The cold junction of the thermocouple was placed in melting ice.

After a number of tests had been run, it was considered possible for the conduction of the brass thermometer cup to lower the temperature as given by the thermocouple. In order to test this, a mercury thermometer, and later a Hoskins thermocouple, the latter having been compared with the platinum-rhodium couple, were placed in direct contact with the gases. The producer was then run at different capacities, in order to produce different temperatures in the leaving gases, and simultaneous readings of the Hoskins couple exposed to the gases and of the platinum-rhodium couple protected by the thermometer cup and quartz glass tube, were taken. The curve of Fig. 7 was plotted from the data taken on this experiment. The results were somewhat startling.

As the points fell on practically a straight line, there could be no doubt of the effect of the thermometer cup and the quartz glass tube upon the reading of the thermocouple. The temperatures used in the computations were taken in degrees F. from curve 1 of Fig. 7. The actual readings of the platinum-rhodium couple in degrees Centigrade were plotted as abscissas while the corrected readings in degrees F. were plotted as ordinates. Curve 2 of Fig. 7 is a curve showing the relation between degrees Centigrade and degrees Fahrenheit, and was used in transferring temperature from one scale to the other. The corrected average temperature of the gases leaving the producer, in degrees F., is given in Table 5, item 39.

(11) *Temperatures in the Fuel Bed.*—The temperatures in the fuel bed were obtained on several tests by the use of a platinum-rhodium thermocouple and the Siemens and Halske galvanometer. The hot junction of this couple was placed in a ¼-in-


TEMPERATURE DEGREES FAHRENHEIT
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TEMPERATURE DEGREES CENTIGRADE
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400
500
600
700
II CENTIGRADE DEGREES TO FAHRENHEIT DEGREES
I CORRECTION CURVE PT.-RH. COUPLE

FIG. 7

quartz glass tube about 30 in. long, the quartz tube in turn being placed in a $\frac{3}{8}$-in. iron pipe closed and pointed at one end. The pipe containing the hot junction of the thermocouple was then

TABLE 3

Temperature Through Fuel Bed

Test No.	Time	Zone No.*	Temperatures °F .		
			3 inches from near Side of Lining	At Center	3 inches from far Side of Lining
6	6:40 to 6:50	1	1652	1535	1688
		2	2192	1742	....
		3	....	1782	1922
		4	1832	2282	1048
8	1:52 to 2:20	1	2490	2120	2060
		2	1832	2247	2327
		3	....	2282	1975
		4	....	1724	1922
18	11:30 to 12:40	1			2300†
		2	2350	2163	2250
		3	....	2380	2375
19	11:00 to 11:40	1	2000	1970	1950
		2	2300	2160	2300†
		3	2350	2137	2250
21	11:45 to 12:30	1	700	575	375
		2	1560	1550	1475
		3	2075	1875	1725
25	10:05 to 10:55	1	2100	2037	2025
		2	2350	2225	2275
		3	....	2200	2400
27	10:30 to 11:40	1	1075	1075	1650
		2	1580	2000	2000
23	8:43 to 9:35	1	1237	1610	1625
		2	1500	1812	1900
		3	1600	2037	2225
	3:50	1	1950	1675	1700

*Zone No. 1 = 24 in. above grate.
Zone No. 2 = 18 in. above grate.
Zone No. 3 = 10 in. above grate.
Zone No. 4 = 3 in. above grate.
† indicates that the temperature rose above the softening point of the thermocouple and hence was not obtained.

inserted in holes drilled through the walls of the producer. These were drilled one above the other at 3 in., 10 in., 18 in. and 24 in., respectively, from the grate. Temperatures were taken first through the upper hole, at 3 in. from the near side, at the center and 3 in. from the far side, and in the same positions through each of the other holes.

The platinum-rhodium couple did not prove entirely satisfactory for this work, on account of the difficulty in properly protecting it. When the end of the iron pipe entered a zone where clinker was forming and where there were large holes in the fuel bed through which the air was passing, the temperature rose rapidly and either melted off the end of the pipe or bent it so that it was difficult to remove. After losing several inches from the end of the couple and at the same time contaminating it with the gases from the fuel bed, it was decided to try the Hoskins couples. These are a patented thermocouple of considerable mechanical strength, supposed to be unaffected by furnace gases. The composition of the elements forming the couple is not known. The temperature as indicated by the Hoskins couple is read in degrees Fahrenheit direct from the galvanometer provided. This latter reads to 2500° F. It was found, however, that the couples softened at 2250° F., so that it was not possible to go higher than this. These couples are much heavier than the platinum-rhodium couples, and, according to the manufacturers' statement, could be used in the fuel bed without protection. It was found better, however, to enclose them in iron pipes. The temperature n the fuel bed was taken by means of the Hoskins couple from test 18 to test 31. The temperature so observed will be found in Table 3.

(12) *Measuring the Water Used.*—The water fed to the producer from the vaporizer in tests 2 to 8 inclusive and in tests 31, 32 and 38, was obtained by weighing a tank filled with water which rested on platform scales. This tank was placed on the charging platform above the producer. The water was allowed to drip from the tank into the vaporizer. A constant level was maintained on the vaporizer by allowing the water to overflow into a second tank resting also on platform scales. The weight of water fed to the producer was therefore the difference between the weight of the supply tank and the weight of the overflow tank.

In all other tests the vaporizer was blanked off and a steam jet supplied the moisture necessary in the operation of the producer. The jet passed through a small thin plate orifice. The pressure on the orifice in pounds per square inch was taken, and after the test, the orifice was calibrated under the pressure observed on the test. The jet was located beneath the grate and in the center of the ashpit. Any steam condensing in the ashpit from the jet overflowed, and was collected and weighed at the end of the test. The weight of this overflow was small and in a number of tests there was no overflow. The weight of water supplied to the producer was obtained from the pressure on the orifice and the calibration at this pressure. The actual weight that entered the fuel from the steam jet was the difference between the above weight and the weight of the overflow. The former quantity is given under item 79A of Table 5. Besides the water from the vaporizer, there is also the moisture carried in by air and by the coal. These weights are given under items 79B and 79C respectively. The computations for the weight of moisture carried in by the air are given in the Appendix, page 87, items 104 and 105.

The water supplied to the first scrubber was measured by a water meter, which was calibrated before the test. The water used by the second scrubber was not considered, as this was only incidental to the method of testing.

(13) *Weight of Moisture in the Gas.*—The weight of moisture in the gas leaving the producer was determined by two separate methods. First, by drawing a sample of gas through a calcium chloride tube, by means of a water aspirator. A small wet gas meter was placed between the calcium chloride tube and the aspirator from which the volume of gas was determined. The moisture accumulating in the tube was weighed. From the data obtained from this apparatus, the per cent of moisture in the gas leaving the producer was determined. In the second method, the weight of water decomposed in the fuel bed was computed from the analysis of the gases, Appendix, page 87, item 86. This weight is given in Table 5, item 80. The total weight of moisture entering the producer is given under item 79; the difference between the two items gives the weight of water in the producer gas. It is believed that the percentage of moisture in the gas based on this determination is the more accurate.

(14) *Ash and Refuse.*—The ash and refuse taken from the grate during the test, together with that resulting from the cleaning at the end of the test, were collected and weighed. The whole was then quartered and sampled, and the sample sent to the chemist for analysis. The weight of carbon lost through the grate was determined from the total weight of ash and refuse and the per cent of carbon as given in the chemist's report. It will be noted from item 49 of Table 5 that in several of the tests, the percentage of ash and refuse obtained is lower than the percentage of ash in the coal as given by the chemist's report, item 53. This is due to the impossibility of removing all of the ash from the fuel bed at the close of the test. This ash may have been either in the form of clinker which accumulated on the lining of the producer, or in the form of a soft white powder which packed into the interstices around the coal. The failure to obtain the correct or true weight of ash formed during the test, as explained under the measurement of the weight of coal fired, is not of great importance, as the weight of ash and refuse is used only for the determination of the weight of carbon lost through the grate.

In a number of tests, ultimate analyses were made on the ash and refuse for the purpose of investigating the formation of clinker. These analyses are given under item 63 of Table 5. The percentages are based on the weight of "earthy matter."

(15) *Soot and Tar.*—The amount of soot and tar escaping the scrubber of the hard coal producer is, under normal conditions, very small, and as a rule very little trouble results from their presence. No attempt was made in the tests to determine their weight. After about 600 hours' operation and the generation of about 2 000 000 cu. ft. of gas, the horizontal pipe leading from the wet scrubber to the steam ejector was taken down and examined. It was found to contain a coating of tar and soot less than $\frac{1}{32}$ in. thick on the inner wall. As this period of time would correspond to about two months, running at 12 hours per day, it may be assumed that the scrubber capacity is sufficient.

(16) *Stand-by Losses.*—Small producers of the present type are usually designed for intermittent operation and are not well adapted for continuous runs of longer than twelve hours' duration. This is a disadvantage due to the size of the producer, and results from the inability to thoroughly clean the ash from the fuel bed

without lowering the heating value of the gas to such an extent as to interfere with the operation of the engine. In larger producers the volume of gas generated per unit of time is large: the fuel bed is also large, so that the air admitted in poking and cleaning the fire represents a very small percentage of the gas volume generated. Also, the proportion of the fuel bed disturbed at one time is relatively small compared with the total area of the fuel bed. On account of these conditions, the fuel bed of a large producer may be cleaned and poked without seriously affecting the quality of the gas, so that continuous operation is a comparatively easy matter, while with a smaller producer of the same type continuous operation would be impossible.

The specifications accompanying the present producer state that it is to be used for runs of not greater than 12 hours. Consequently, there will be a stand-by of twelve hours' duration if the engine is being operated twelve hours per day. As this is the condition that maintains in commercial work, it is of importance that the losses due to this stand-by be known.

In order to determine these, a stand-by test of 120 hours starting October 11, 1909 at 10:20 a. m. and running until October 16, at 10:20 a. m. was made. The test was started by running the producer until a normal fuel bed was obtained, and then cleaning the fires and filling the producer with coal. The producer was then closed down, and at the end of twelve hours, the fires were cleaned, and the producer operated until the gas would burn at the try cock. The producer was again closed down for twelve hours, and at the end of the period, the fires were again cleaned and the above cycle of operations repeated. During the 120-hr. run, the fires were cleaned and the gas producer operated until a working gas was obtained ten times, that is, once every twelve hours.

At the close of the test, 280 lb. of coal were required to fill the producer. This represents a stand-by loss of 28 lb. of dry coal per twelve hours. If we consider that 500 lb. of coal per 12-hr. run is about the normal capacity of the producer, the stand-by loss represents about 5.5 per cent of the dry coal fired or about 6 per cent of the combustible. The results of the stand-by test are given in Table 4.

The stand-by loss is divided into four parts; (1) that which is lost through the grate due to cleaning the fire; (2) that which is lost through the poor gas generated from the time the blower is started until the gas is sufficiently rich for use in the engine; (3) that which is lost in the small quantity of gas generated during the actual stand-by period; and (4) that which is due to the conduction and radiation of heat from the producer during this stand-by period.

The loss through the grate may be obtained from the data of Table 4. It was not deemed sufficiently important to separate the second and third losses, and it was not possible to obtain the fourth other than approximately.

TABLE 4

Results of Test for the Determination of Stand-by Losses.
Time of start, 10:20 a. m., October 11, 1909.
Time of stop, 10:20 a. m., October 16, 1909.
Duration of the trial, 120 hours.
Producer started 10 times.
Fuel used, anthracite.
Commercial name, Scranton pea.

FUEL

1	Size and condition	Pea clean
2	Weight of coal as fired, lb	288.0
3	Percentage of moisture in coal	2.92
4	Total weight of dry coal fired, lb	280.0
5	Total ash and refuse, lb	51.0
6	Quality of ash and refuse	
7	Total weight of combustible, lb	212.0
8	Percentage of ash and refuse in dry coal, per cent	18.2

ULTIMATE ANALYSIS OF DRY COAL

9	Carbon (O)	per cent	79.83
10	Hydrogen (H_2)	"	2.59
11	Oxygen (O_2)	"	2.20
12	Nitrogen (N)	"	.82
13	Sulphur (S)	"	1.39
14	Ash	"	13.17
15	Moisture in sample of coal as received	"	2.92

ANALYSIS OF DRY ASH AND REFUSE

16	Carbon, per cent	43.00
17	Earthy matter, per cent	57.00

FUEL PER HOUR

18	Dry coal fired per hour, lb	2.34
19	Combustible consumed per hour, lb	1.77
20	Dry coal per sq. ft. of grate area per hour, lb	1.40
21	Combustible per sq. ft. of grate area per hour, lb	1.06
22	Dry coal per sq. ft. of fuel bed per hour, lb	1.25
23	Combustible per sq. ft. of fuel bed per hour, lb	.94

10. *Duration of the Tests.*—Owing to the conditions under which the tests were run, it was not possible to make runs of longer than

twelve hours' duration. Fortunately, however, it will be found, from the discussion under the measurement of the weight of coal fired, that with the exception of the very low capacity tests, runs of this length were sufficient to reduce the probable error in determining the weight of coal fired, to about two or three per cent. It has also been noted that owing to the small size of the producer, tests of longer duration are hardly practicable. This is due to the fact that toward the end of all 12-hr. tests, the accumulation of ash in the fuel bed necessitated such thorough cleaning as to seriously interfere with the uniformity of the heating value of the gas. As the producer is provided with a plain bar grate and all the cleaning must be accomplished by opening the ash doors, this particular type of producer, while operating with a fair degree of satisfaction on runs of not greater than 12 or 15 hours' duration, would not be satisfactory for continuous runs of longer duration. This has been recognized by the builders and was so stated in their specifications.

As the object of these tests was to show the actual operating efficiency of the producer, and as under these conditions the producer would not be run for greater than 12 continuous hours, the present tests have been made to conform to those conditions as nearly as possible. The producer necessarily shows a lower efficiency under these conditions than it would under continuous operations. The reason for this lower efficiency is the stand-by losses. These are composed of four separate losses, as has been pointed out. One of these affects directly the results of the tests. This is the loss due to the radiation and conduction of heat from the fuel bed and producer during the stand-by period. It results from the fact that at the close of a run or test of 12 hours' duration, the fires are cleaned, and in cleaning, a large quantity of incandescent ash and carbon is removed from the fuel bed; this mass is replaced by the green fuel from above so that after cleaning, the temperature of the fuel bed is low and remains so until the starting of the new run or test. The producer lining, shell water jacket and water also lose heat during this stand-by period. On again starting the producer at the beginning of a test, the fire is blown until a gas sufficiently rich for operation is produced, and the test is then started. The average temperature of the fuel bed and producer has not reached the temperature of normal operation. At the close of the test, this temperature has been reached, consequently, a large quantity of heat must be used in raising the average temperature of the producer at the begining of the test to the average temperature at the close of the test. In the case of the present test, the producer was not in operation continuously and it was not always possible to bank the fires from one test to another, for this reason. The fire in the producer was usually started on an average of about two hours before the test began. Gas sufficiently rich for operating could be obtained in about thirty minutes from the time of starting the fires, the additional time permitting the producer to warm up. Notwithstanding this, however, the difference between the average temperature of the producer at the close and the average temperature on the starting of the test may have been anywhere from 200 to 700°F. The magnitude of this loss and the extent to which it probably affected the heat balance of tests may be best illustrated by an example. Assume that the average temperature of the producer at the starting of the test is 1000°F., assume that the average temperature at the close of the test is 1600°F. The difference in temperature is about 600° F. The total mass in the present case represents about 1200 lb. If we assume the specific heat to be .25, the total heat that has been lost during a run in bringing the average temperature of the producer at the start of the test to the average temperature at the close of the test is 1200 × 600 × $\frac{1}{4}$ = 180 000 B. t. u. On the high capacity tests this loss is a very small percentage of the

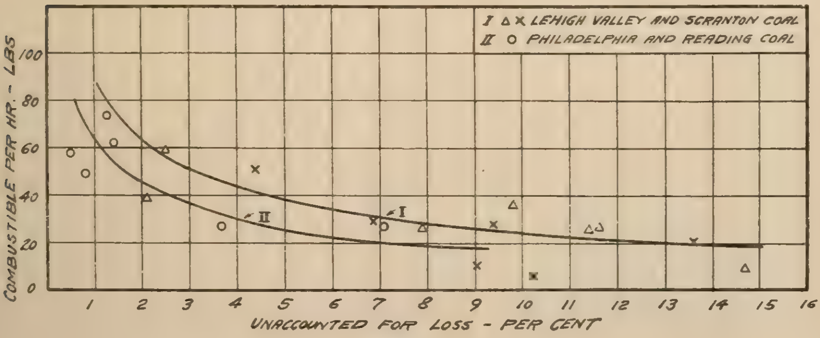

FIG. 8

total heating value of the fuel, on the lower capacity tests it is a very large percentage. Suppose 150 lb. of coal are fired on one of the light load tests. The heating value of this coal is 12 500 B. t. u. per lb. The total heat supplied to the producer is 12 500 × 150 = 1 870 000. The per cent loss due to the above is, therefore, $\frac{180\,000}{1\,870\,000}$ = 9.6 per cent. As this loss is largely inherent in the operation of the producer, it was deemed advisable to indicate its magnitude and not eliminate it from the results of the tests.

The curves of Fig. 8 have been plotted from the results of all tests and show approximately how this loss varies with the weight of combustible gasified. These curves would also seem to indicate that the radiation and conduction losses from the producer are between 0.7 and 1 per cent, since the curves become parallel to the Y axis in this neighborhood.

11. *Forms for Computation and Discussion of Items of Table* 5.—Three forms have been drawn up for use in making the computations for the tests. Table 5 is made according to form 1, which is used for the presentation of the results of the test; a blank of this form is shown in the Appendix. Form 2 was made to include all items involved in computation of the results and form 3 contains the derivation and discussion of the formulas used. In working up the tests, the average total quantities are taken from the original log sheets, corrected and placed on form 2. The item number of form 3 refers to the item number of this form. The results are computed in the order of form 3 and placed on form 2, from which the results compiled are placed on form 1. In order to find the formulas used in computing any result given in Table 5, this result must be found on form 2, and the item number corresponding to this result on form 2 corresponds to the item number of form 3 under which the formulas used will be found. For example, suppose that it is desired to find the formulas used in computing the cold gas efficiency as given in Table 5. Referring to form 2, Appendix, it will be found that the cold gas efficiency on this form appears under item 155. By referring now to item 155 of form 3, the formulas required will be found.

One of the important considerations in developing the forms has been to include such items as will give proof of the accuracy of the work, and that will point out the nature of any serious errors that are always likely to creep into experimental work.

The gas volume has been determined from a previously calibrated meter. This volume has been checked in most tests by computing the volume of gas from the weight of coal consumed, the analysis of the coal and the analysis of the gases. If the volume of gas as given by the meter is not more than 5 per cent lower than the computed volume, it is assumed that the gas volume as determined by the meter is correct. If there is a greater variation than this, the error may be either in the weighing or estimation of the coal, in the meter, or in the analysis and sampling of the gas. If the heating value of the gas as computed from the analysis checks the heating value as given by the Junker calorimeter, the conclusion is that the error is on the side of the coal. If the volumes check, and also the heating value of the gas as determined from the analysis and from the calorimeter, the conclusions are that the only serious error that can exist must be in the measurement of the temperature of the gases leaving the producer. Even this temperature may be checked by computing from the weight of scrubber water, item 90, and from the rise in temperature of the scrubber water, the heat lost to this by the gases leaving the producer.

The word combustible has been used in these tests according to the definition appearing in the Appendix, forms 2 and 3, item 54. As this word has been used in an arbitrary sense by a number of writers and also by the engineering societies, its definition is of considerable importance.

The calorific value of the gas is given under items 104 to 105 Table 5 and form 1. The high value is defined as the total heat given out by a cubic foot of gas at 62° and 30 in. of mercury when it is burned in oxygen and the products resulting from combustion are brought back to 62° F. The net or effective value is equal to the "high" value minus the latent heat contained in the water that is formed by the combustion of the hydrogen. It will be seen that if the gas is used under such conditions that it is exhausted at a temperature greater than 212° F., this latent heat is of no use in the cycle of operation. This is the case where the gases are used either in the gas engine or for firing boilers. The low or effective value is given under item 104a which is computed from the analyses of the gases.

Under item 126 is given *the grate efficiency.* This is a very variable quantity and depends upon the fireman, the size of coal fired, the amount of ash contained in the coal and upon the number of times the fires require poking and cleaning. With large producers and careful firing, this item should not be less than 95 per cent. With small producers of the type used in this work 95 per cent is an excellent result. With the coals used on the tests, from 80 to 90 per cent is probably a fair value for this item.

The hot gas efficiency based on the high heating value of the gas is given under item 127. This value varies from 100 per cent principally by the percentage of the radiation, conduction and unaccounted-for loss and by the percentage of the heating value of the fuel lost through the grate. The hot gas efficiency is of no special importance where the gases are used for power purposes.

The cold gas efficiency based on the high heating value of the gas appears under item 128, again under item 128a, based on the low or effective heating value of the gas. This efficiency for both high and effective heating values has also been based on 100 per cent grate efficiency or upon combustible and is given in items 128c and 128d. This latter value has been computed in order to show certain relations that are independent of the grate efficiency.

12. Under *unaccounted-for loss* may be grouped the following losses.

a. Radiation
b. Conduction
c. Loss due to tar and soot and to the absorption of the gases by the scrubber water.
d. Loss due to difference in temperature between the initial and final conditions of the fuel bed.
e. Minor losses which may be either positive or negative such as the loss due to the sensible heat in the fuel and in the ash.
f. Experimental errors made on the tests.

TABLE NO. 5 GENERAL RESULTS OF TESTS

No. of test	6	8	7	4	2	3	21	20	19	15	16	18	17	26	32	27	31	24	23	33	25	30	29	28	*14
Date of test	12-30-08	1-2-09	1-1-09	12-21-08	12-15-08	[illegible]	[illegible]	5-5-09	4-10-09	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
Duration of test	9 hr	9 hr.	9 hr	8 hr	7 hr	12 hr	12 hr	12 hr	12 hr	15 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	12 hr	8 hr
Kind of fuel used	Philadelphia and Reading Chestnut						Lehigh Valley Chestnut							Scranton Pea						Gas-House Coke					P and R
Fuel	Chestnut—dirty						Chestnut—clean							Pea—clean						Irregular—dirty					

* Test run to determine the effect of [illegible] stone as a flux for the clinker [illegible] limestone charged with [illegible] coal

NOTE—In cases where [illegible] the vaporizer was not used, the steam being supplied from the main.

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PART III. RESULTS OF THE TESTS

13. *General Discussion of the Tests.*

(1) *Tests with Philadelphia and Reading Anthracite.*—Tests 2 to 8 inclusive, Table 5, were run on Philadelphia and Reading anthracite coal, chestnut size. This was in appearance a dirty anthracite, probably 50 per cent of which would pass over a $\frac{3}{4}$-in. mesh and through a $1\frac{1}{2}$-in. mesh screen, while 25 per cent would pass a $\frac{5}{16}$-in. mesh.

The average percentage of ash as taken from the analysis of the dry coal was approximately 15 per cent. Analysis of this ash given in Table 5, under test 4, shows that 90 per cent by weight was composed of the oxides of silicon, iron, and aluminum; the remaining 10 per cent consists of the oxides of calcium and magnesium, which in combination with the above, are sufficient to form a semi-fusible, viscous ash at the temperature obtaining in the fuel bed of the gas producer.

On the tests it was found that the clinker began to form at very low temperatures, and by the time the temperature of the fuel bed rose to an efficient working temperature, the formation of clinker on the lining of the producer was so rapid as to interfere with the quality of the gas. By the use of larger quantities of steam, or at lower capacities, it would doubtless be possible to use this coal in a producer of larger size with satisfactory results. With a producer of the size and type used on the tests, satisfactory operation is practically impossible.

This series of tests is of little value except to illustrate the effect of a fuel containing a fusible ash upon the operation of a small producer. Owing to the formation of clinker on the sides of the producer, the fuel bed is greatly reduced in area, and there is the tendency for the formation of holes which permit the air to rush through, resulting in high temperature and poor gas. Owing to the frequency of poking, large quantities of incandescent coal drop into the ashpit, thus reducing the grate efficiency. Fig. 9 is a graphical log of test 6.

(2) *Tests with Lehigh Valley Anthracite.*—Tests 15 to 21 inclusive were run on Lehigh Valley anthracite, chestnut size. This was extra large chestnut and clean in appearance. Ninety per cent of this coal would pass through a 2-in. mesh, while all of it would practically pass over a $1\frac{1}{4}$-in. mesh. The average percent-

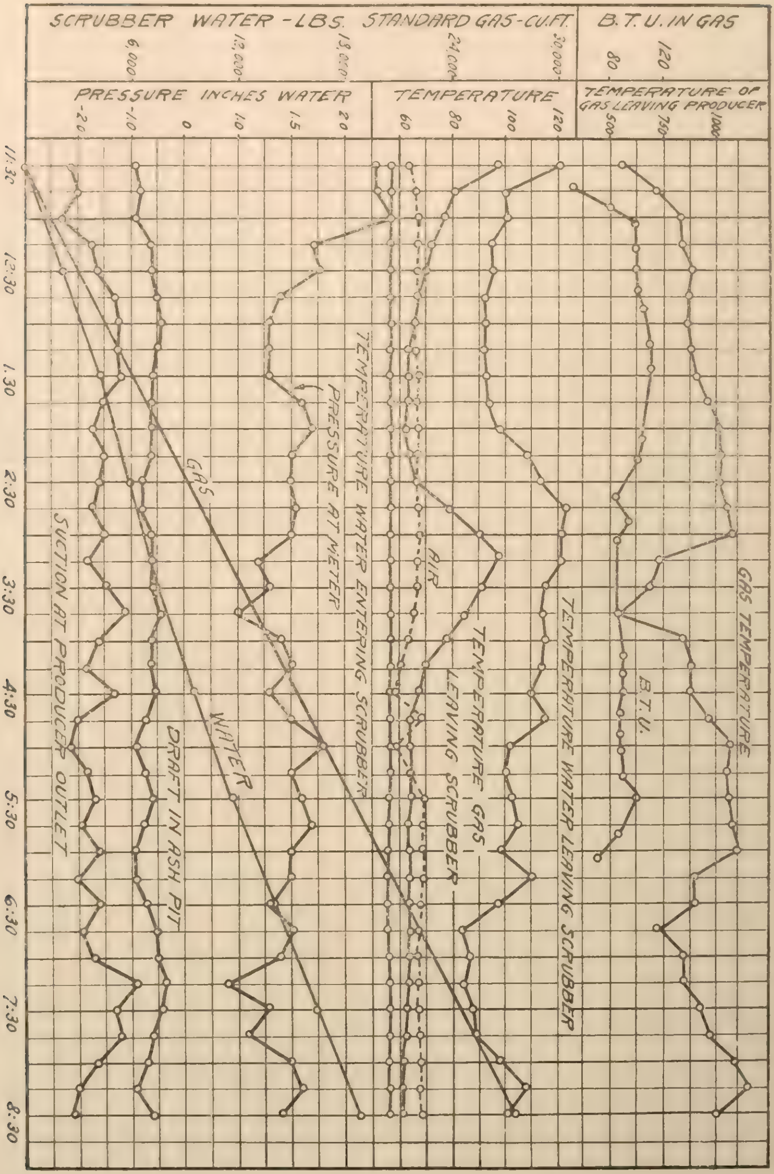

Fig. 9
Graphical Log Test No. 6

age of ash as taken from the analysis was approximately 15 per cent.

The ash taken from the producer at the close of the tests was soft and white and contained small pieces of clinker. There was practically no tendency for the formation of large clinker arching across the fuel bed at the higher temperature.

The conditions that maintained in this series were fairly uniform, considering the size of the producer. Fig. 10 and 11 show the graphical logs for tests 17 and 20, respectively.

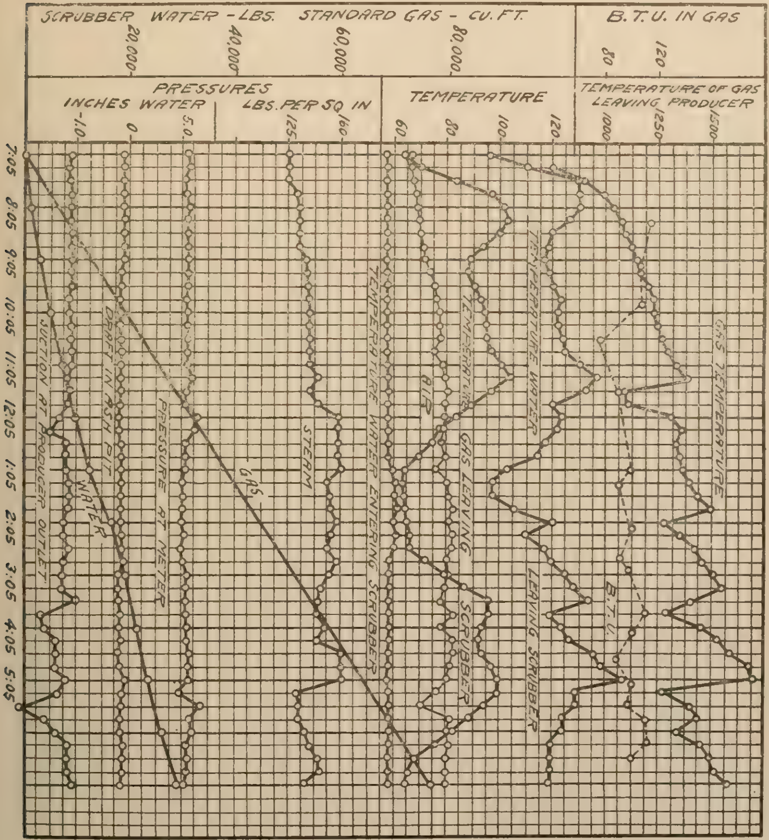

FIG. 10
GRAPHICAL LOG TEST NO. 17

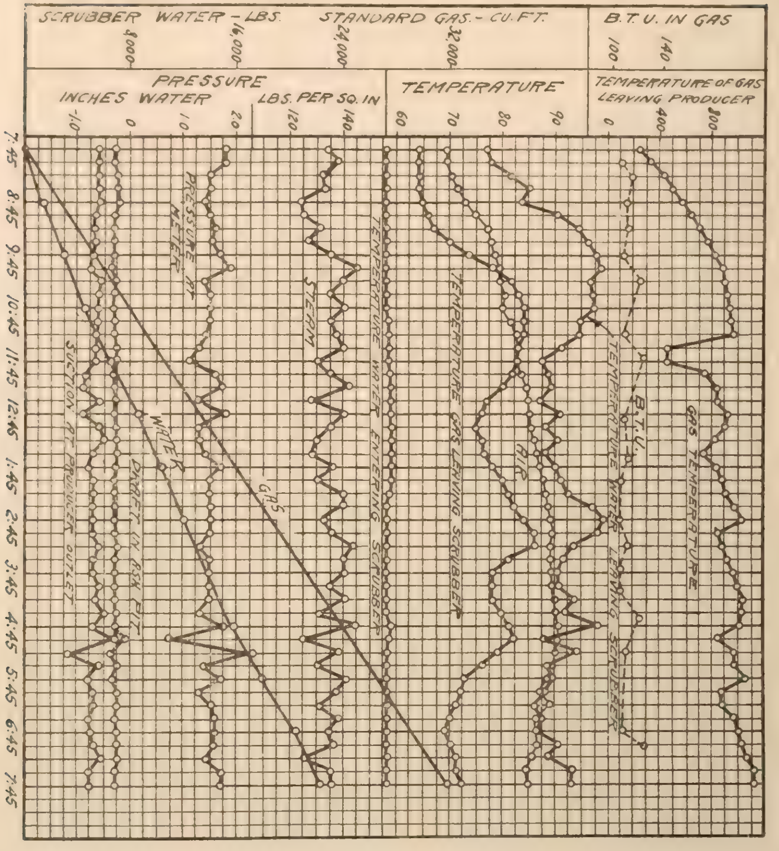

FIG. 11
GRAPHICAL LOG TEST NO. 20

The average heating value of the gas for all of the tests was 110. The variation of the heating value on all of the tests, while probably sufficient to cause considerable trouble in the operation of an engine driving electric lighting machinery, would not interfere seriously with the operation of an engine where close regulation was not a prime consideration.

(3) *Tests with Scranton Pea Coal.*—Tests 23 to 27, including tests 31, 32 and 38 were run on Scranton pea anthracite. This coal was clean in appearance and like the other two coals, contained on an average about 15 per cent ash, while the calorific value corresponded to the calorific value of the other coals and

was about 12 800 B. t. u. per lb. of dry coal. It would all pass over a $\frac{3}{8}$-in. mesh and through a $\frac{3}{4}$-in. mesh. Analysis of the ash is given under test 23. This shows it to be an ash consisting of about 96 per cent of the oxides of silicon, iron and aluminum, while the remainder is made up of the oxides of calcium and magnesium, which are present in insufficient quantities to make the ash readily fusible at the temperature maintaining in the fuel bed.

The ash as taken from the fuel bed was soft and fine, mixed with small pieces of clinker. There was very little tendency toward arching and the tests were the most satisfactory of the series. The conditions were as uniform as could be expected,

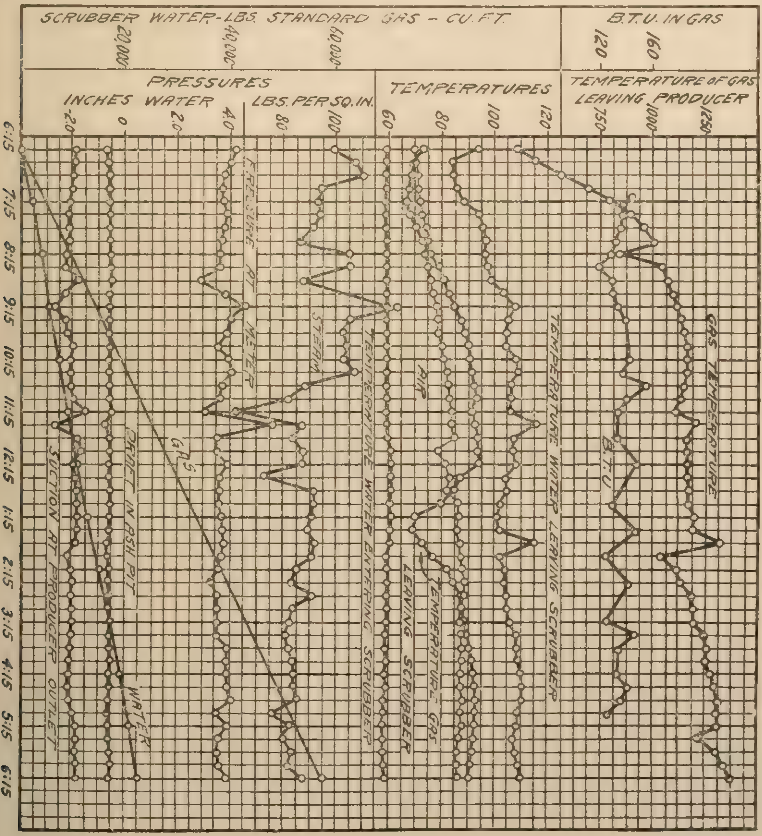

FIG. 12
GRAPHICAL LOG TEST NO. 25

and the efficiency reasonably high. Fig. 12, 13, and 14 are graphical logs from the tests of this coal.

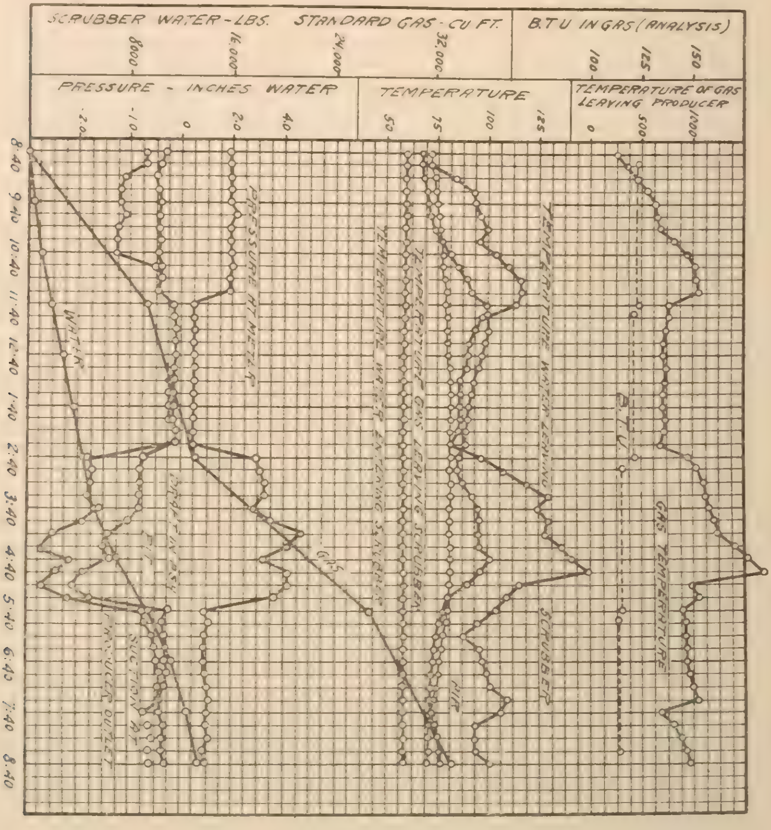

Fig. 13
Graphical Log Test No. 31

Fig. 13 shows the results of test 31, which was a variable load test run for the purpose of showing the effect of suddenly varying the rate of gasification. For the first three hours of the test, the producer was operated at the rate of about 3300 cu. ft. of gas per hour; for the next three hours at the rate of 1155 cu. ft.; for the third 3-hr. period at 4500 cu. ft.; and for the last 3-hr. period at a rate about 2200 cu. ft.

(4) *Tests with Gas House Coke.*—Tests 28, 29, and 30 were run on

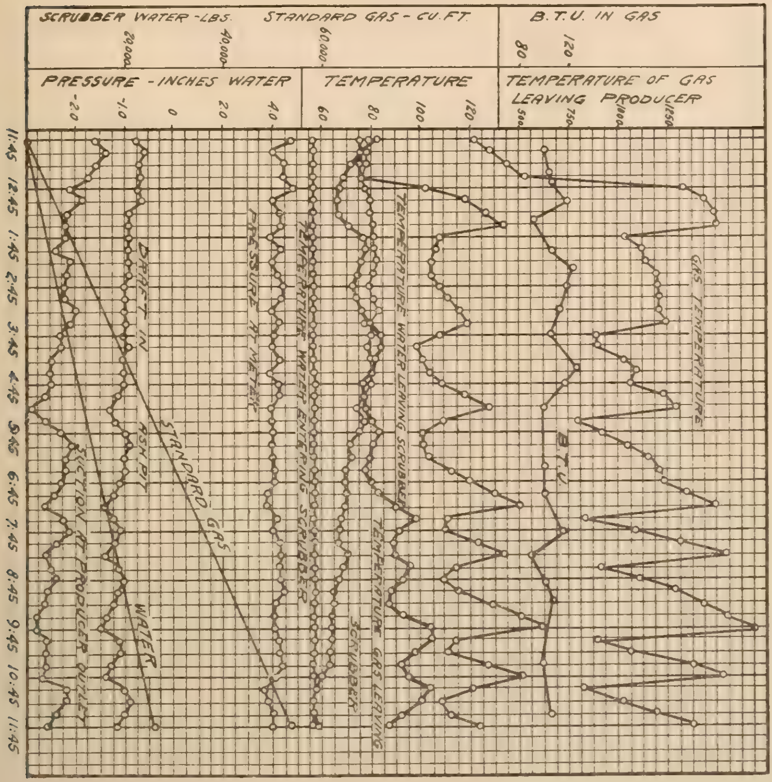

FIG. 14
GRAPHICAL LOG TEST NO. 32

gas house coke containing about 14 per cent ash. The appearance was dirty and the size very irregular. Thirty per cent of this would doubtless pass a $\frac{5}{16}$-in. mesh, 50 per cent a 2-in. mesh, while the remainder would pass a 4-in. mesh.

There was some trouble due to the formation of clinker, and considerable trouble caused by a tendency of the coke to pack in the hopper, thus preventing its descent into the fuel bed. In spite of this, the conditions that maintained on the tests were fairly uniform, and it is believed that very little difficulty would be experienced in using a coke of uniform size and quality in this type of producer. Fig. 15 is a graphical log plotted from the results of test 30.

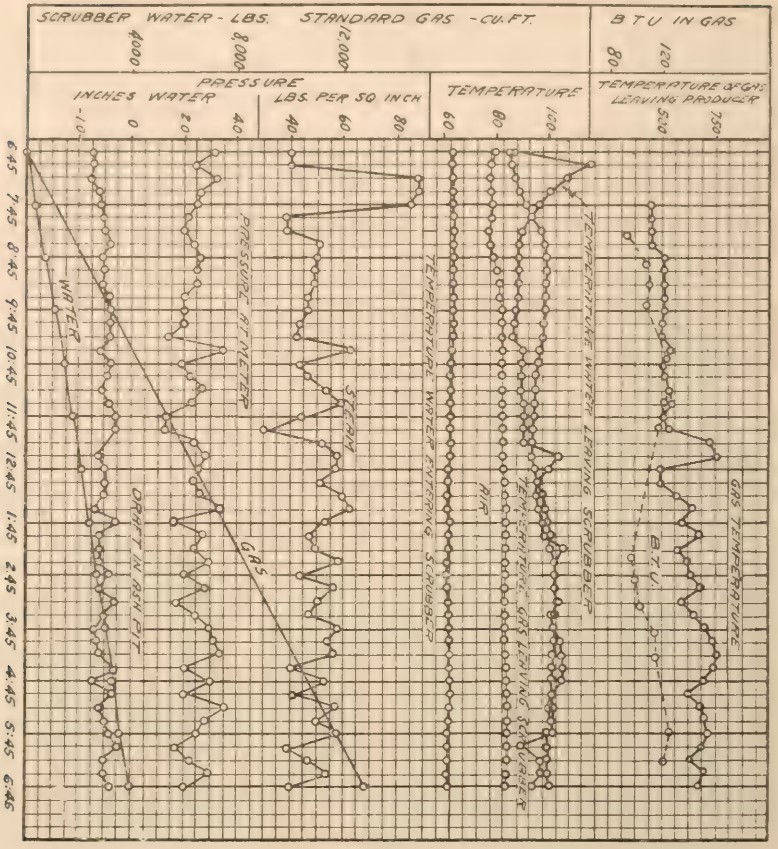

FIG. 15
GRAPHICAL LOG TEST NO. 30

(5.) *Test No. 14.*—Test 14 was a special test run on the Philadelphia and Reading coal to determine if the ash in this coal could not be made sufficiently fusible to run from the fuel bed. This test will be discussed under the formation of clinker.

14. *Percentage of CO_2.*—The percentage of CO_2 in the gas is ordinarily considered to be a measure of the efficiency of a producer, low percentages indicating high efficiency while high percentages indicate the reverse in efficiency. This is to a certain extent true in most producer practice, but it is not necessarily so, as the following will illustrate. If the heat resulting from the formation of CO_2 is utilized in the decomposition of the vapor of water for

the production of H_2, which enriches the gas, then no appreciable loss results. If, however, the CO_2 results through poor operating methods, such as insufficient or improper poking, resulting in holes in the fuel bed and thus permitting the air to rush through, then the presence of a high percentage of CO_2 is indicative of a low efficiency, and this condition will manifest itself in excessively high temperatures of the leaving gases.

This latter condition is illustrated by the results of all the tests on the Philadelphia and Reading coal (see Table 5). The temperatures of the gases leaving the producer, item 39, are excessive, the CO_2 item 117, is high, while the efficiencies, items 126-128d, are low.

The former condition, in which both the CO_2 and the efficiency are high, is illustrated by the tests on the Lehigh and Scranton coals. The principal results of test 23, which have been placed in Table 6 for convenience in discussing, illustrate this condition. The per cent of CO_2 in the gas in this test averaged 10.15. The temperature was, however, low, 800°F., and the efficiency high, thus showing that the fuel bed was in good condition but at so low a temperature, due to the use of a large amount of steam in the air fed to the producer, as to produce a predominance of the reaction 3 which has been discussed in page 6 of the theory. Test 24, also placed in Table 6, shows practically the same efficiency but with 4.04 % CO_2 in the gas. This is due to a high temperature in the fuel bed which tends to cause reaction 4 to predominate, resulting in a high percentage of CO, practically the same temperature of the leaving gases as was obtained in test 23, and practically the same efficiency.

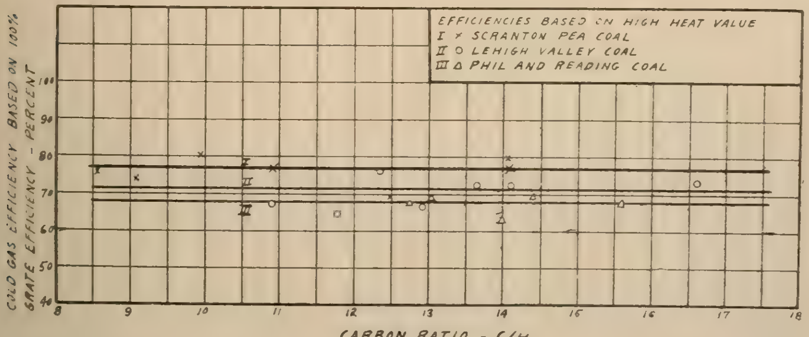

FIG. 16

The curves of Fig. 16, 17, 18, and 19 also show that the per cent of CO_2 has very little effect upon the efficiency of the producer. The theoretical efficiency curve showing the efficiency for different percentages of CO_2 on the assumption that the heat liberated through the formation of the CO_2 is lost in sensible heat, has been drawn and is shown in Fig. 19. The difference between the actual curves and the theoretical curve is a measure of the amount of heat conserved through the decomposition of water vapor.

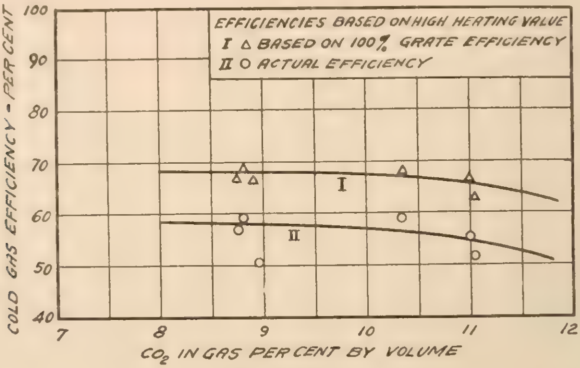

FIG. 17

CO_2—EFFICIENCY CURVES FOR PHILA. AND READING COAL

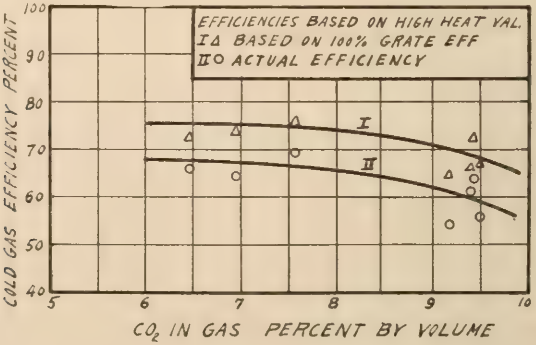

FIG. 18

CO_2—EFFICIENCY CURVES FOR LEHIGH VALLEY COAL

The efficiency curves on this figure have been affected by the unaccounted-for loss which has been discussed on page 37, and has resulted in lowering the points in the low capacity tests to such an extent as to make these curves concave to the axis.

The curves of Fig. 18 are also affected by the unaccounted-for loss, but in this case the gas from the low capacity tests ran high in CO_2, owing to the use of a large quantity of steam, which resulted in causing the curves to drop off at the high percentage of CO_2 faster than they otherwise would.

The curves of Fig. 16, showing the relation between the efficiency and the carbon ratio, or the ratio of the weight of carbon to the weight of hydrogen in the producer gas, illustrate the point under discussion, showing that the efficiency of the producer is practically independent of the amount of hydrogen in the gas, consequently, of the per cent of CO_2 in the gas. Fig. 20 shows how the per cent of CO_2 in the gas depends upon the weight of water decomposed per pound of combustible.

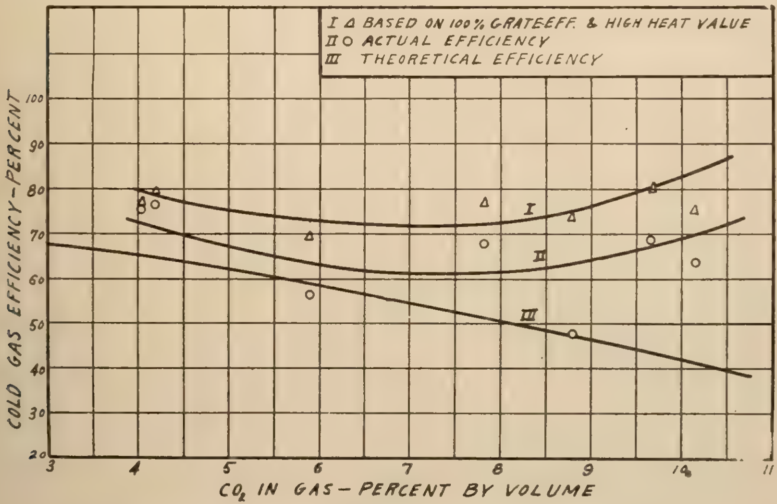

FIG. 19

CO_2—EFFICIENCY CURVES FOR SCRANTON PEA COAL

15. *The Effect of the Size of the Fuel.*—The determination of the effect of the size of the fuel upon the composition of producer gas and also upon the efficiency of the product is complicated by the difference in operating conditions that may not be recognized and upon the difference in the surface effect of the incandescent fuel upon, first, the reactions of oxygen and hydrogen and later on the reaction of CO_2 on this fuel. The exact nature of the surface

effect, which is called "catalytic action", and under which a number of different effects are grouped, is very little understood at the present time. It probably results from chemical reaction on the surface of solids in contact with gases, which depend upon the state or form of the solid, such as charcoal, lampblack, coke or anthracite coal, and in the case of these fuels upon any impurities contained in them. The result of catalysis is to hasten any reaction that may be taking place between the solids and the gases in contact with them, or between the gases themselves. It is believed, however, that the ultimate extent or that the equilibrium of the reaction is not affected.

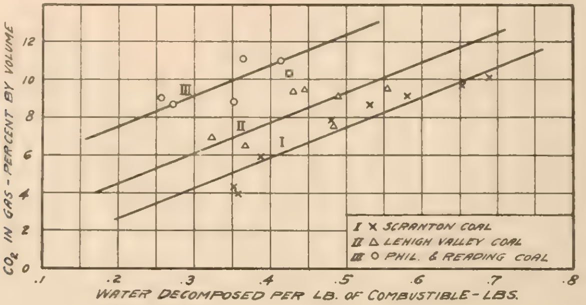

FIG. 20

It is probable that the difference in the catalytic action of different fuels, such as were enumerated above, may be very great. It is also probable that this effect is not so great between the same types of fuel, as, for example, between two grades of anthracite, or between two grades of coke. The ash in the fuel, as above noted, may also have a decided effect upon this action.

The difference in the catalytic action of charcoal, coke and anthracite is illustrated by Fig. 21, 22, and 23, which are taken from Dr. Clement's work, "On the Rate of Formation of CO in Gas Producers".[1] The results were obtained by passing CO_2 over charcoal, coke, and anthracite coal, respectively. In each case the material was ground to a certain size, about 0.2 in. on the side, and placed in a porcelain tube that could be maintained at different temperatures by means of an electric current, traversing a winding of nickel wire placed around the tube. The velocity of the gas, the per cent of CO formed, and the temperature of the tube were the variables. Fig. 21 shows the relation for charcoal between the per cent of CO formed and the velocity of the gas through the tube. Fig. 22 and 23 show the results for coke and anthracite coal, respectively. The velocity, which is the reciprocal of the time of contact, is expressed in feet per second.

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[1] Bull. No. 30, Eng. Exp. Sta., J. K. Clement, L. H. Adams and C. N. Haskins.

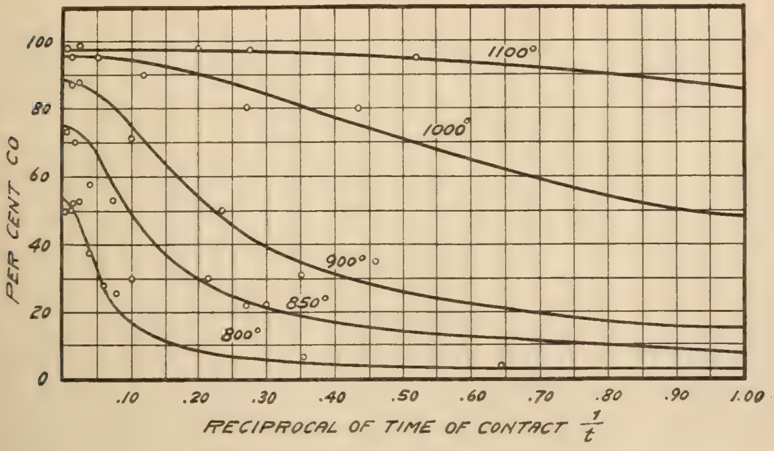

FIG. 21

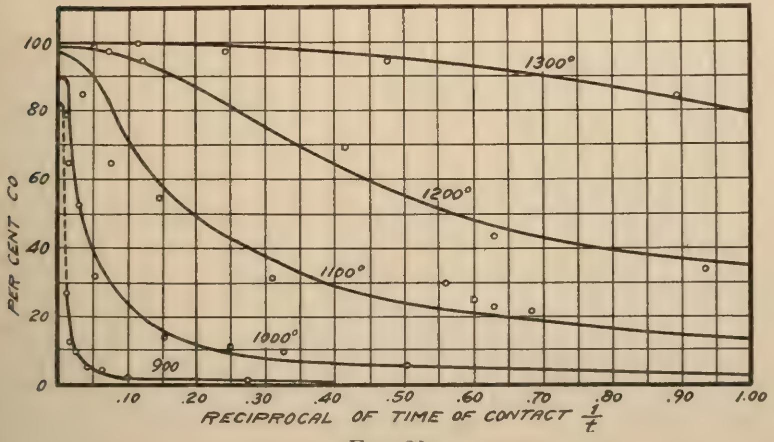

FIG. 22

It will be seen from these figures that the percentage of CO formed by the reaction $CO_2 + C = 2\,CO$, depends upon the temperature, upon the velocity or the reciprocal of the time of contact of gases with fuel, as it has been expressed on the figures, and upon the catalytic action of the fuel. This latter effect is shown by the difference in the per cent of CO formed at the same temperature and under the same velocity for the different fuels used

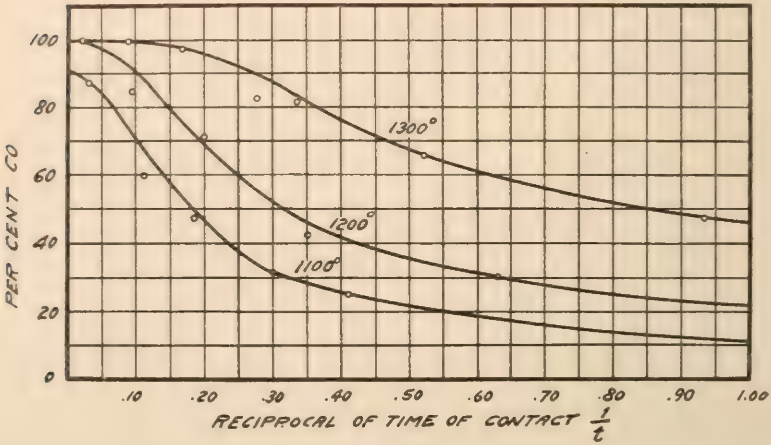

FIG. 23

in the experiments. For charcoal, Fig. 21, the per cent CO for a velocity of 1 ft. per second and a temperature of 1100° C., is 85 per cent, for coke and anthracite the percentages under the same condition, (Fig. 22 and 23) are 9 and 11 respectively. For a temperature of 1300° C. and a velocity of 1 ft. per second the percentages of CO formed are 100, 74, and 45 for the above fuels.

Tests 18, 19, 24 and 25 have been selected as representing most decidedly the effect of the size of the fuel upon the results of the performance of the producer.

Tests 19 and 24 were run under practically the same operating conditions, the only known variable being the size of the fuel, catalytic action excepted. Tests 18 and 25 were also run under essentially the same conditions of operation, with the size of the fuel constituting the only known variable. The principal results of the tests have for convenience been transferred from Table 5 to Table 6.

TABLE 6

1	Test No	19	24	18	25	23
2	Fuel	Lehigh	Scranton	Lehigh	Scranton	Scranton
	Size of fuel, mean diameter in in	1½	½	1½	½	½
3	Capacity in cu. ft. per hr	2730	2941	4545	4795	2980
4	Water decomposed per lb. of combustible	.429	.358	.446	.355	.689
5	Ratio of water decomposed to water supply	.323	.429	.408	.639	.402
6	Per cent CO_2 in the gas	9.4	4.04	9.44	4.2	10.15
7	Per cent H_2 in the gas	9.3	10.80	8.95	10.40	16.96
8	Per cent CO in the gas	17.43	27.3	17.13	27.01	17.39
9	B.t.u. per cu. ft. (High)	106.9	138.2	101.5	138.1	128.8
	(Low or effective)	101	131.6	96.0	131.7	119.4
10	Cold gas efficiency based on 100 per cent grate efficiency (High)	66.9	77.2	72.7	79.9	75.4
	(Effective)	63.3	73.5	68.6	70.3	69.8
11	Per cent loss in sensible heat and in moisture	19.3	12.7	22.7	14.8	15.3
12	Unaccounted-for loss	11.6	9.4	2.1	4.4	6.9
13	Temperature of gases leaving the producer °F	970	809	1201	1108	800
14	Temperature in the fuel bed °F, average	2200		2300	2300	1740
15	Equivalent velocity of gas, feet per second	8.7	3.14	14.46	5.12	2

It will be seen from this table that the per cent of CO_2 for tests on the Lehigh chestnut is about 9.4 per cent, while the per cent CO_2 for the tests on the Scranton pea is about 4 per cent. The heating value of the gas is low in the former tests, as is also the efficiency, while the temperature is a little high. The relative percentage of CO_2 in the gas in the case of these tests is therefore to a certain extent a measure of efficiency of the producer.

The difference in the percentage of CO_2 appearing in the gas from the two coals in the tests under consideration may be due to two causes, viz., (a) the difference in the catalytic action of the two fuels, as above discussed, and (b) the difference in the size of the fuel. In the case of the Scranton pea which, as has been noted, would pass a ¾-in. mesh, the amount of catalytic surface exposed to the action of the gases is much greater than the surface exposed in the case of the Lehigh coal, which will pass over a 1¼-in. mesh and through a 2-in. mesh. The following will illustrate this. The mean diameter of the Scranton coal may be taken as ½ in., the mean diameter of the Lehigh coal may be taken as 1½ in.; since the number of pieces contained in a given volume is inversely as the cube of the mean diameter, and since the surface of each piece is directly as the square of the mean diameter, the ratio of the total surfaces exposed in the fuel bed of the producer will be as 3 and 1 for the Scranton and Lehigh coals, respectively.

Theoretically, the size of the fuel will have no effect upon the velocity of the gas through the fuel bed for the same rate of gasification. This is due to the fact that the number of pieces of coal required to fill the producer varies inversely as the cube of the mean diameter of the pieces, while the volume of each separate piece varies directly as the cube of the mean diameter, the volume of the coal present, and the volume of voids; consequently, the velocity of the gas through the fuel bed is independent of the size of the fuel.

That the voids are, in the case of the two coals, practically of the same volume, has been determined by water displacement. The volume of voids, in the case of the Scranton coal, was found to be 46 per cent of the volume occupied by the coal and 43 per cent for the Lehigh. The velocity of the gases through the fuel bed is therefore practically the same for the two coals for the same temperature and rate of gasification. The amount of draft required to drive the gases through the bed will, however, be different, owing to the increased frictional resistance due to the smaller voids in the Scranton coal.

The purely mechanical effect of the increase in the size of the fuel will be, from the above, if the volume of the fuel bed remain constant, to decrease the area of surface exposed to the gases, to decrease the number of pieces of coal in the path of the gases, and to decrease the drop in the pressure of the gases through the fuel bed.

The decrease in the drop in pressure may be assumed to have no effect upon the composition of the escaping gases within wide limits. The extent of the effect of decreasing the area of the catalytic surface exposed to the gases and of decreasing the number of pieces of coal in the path of the gases upon the composition of the producer gas generated and upon the reaction within the fuel bed is unknown. Dr. Clement's work shows the effec tof the velocity of the gases upon the reaction of CO_2 with incandescent carbon. Our tests No. 24 and 25, Table 6, on the Scranton coal, seem to show the effect of the velocity of the gases on the performance of the producer, tests 18 and 19 also show the same. It will be noted from the above that the increase in the rate of gasification of about 100 per cent has not resulted in any marked change either in the composition, or the heating value of the gases in the case of tests 24 and 25. Tests 18 and 19 on the Lehigh coal show a slightly greater difference, and it is believed this is due to the lesser catalytic surface which results in a shorter time of contact of gases with fuel. From these tests and the curves of Fig. 21, 22 and 23, and beyond the velocity of 1 ft. per second (the curves are drawn only to this velocity) which corresponds to the velocity of the gases obtaining in the fuel bed of the producer, it will be seen that the effect of the velocity on the reaction of CO_2 on carbon is small.

As before pointed out, the difference in the velocity of gases through the fuel bed for the same rates of gasification for the two fuels, if any, must be small. The great difference in the CO_2 or in the composition of the gases from the two fuels for the same rates of gasification as above noted, must therefore be due to one or more of the following causes: (1) to the greater catalytic surface exposed to the gases as in the case of the Scranton coal; (2) to the greater number of CO_2 molecules coming in contact with the coal, due to the greater number of pieces; (3) to the smaller size of the voids in this coal which would tend to cause a more intimate mixture of gases with the coal; or (4) to a greater catalytic effect. It has been remarked that it is probable that the difference in catalytic action between two grades of anthracite coal is small. We will therefore assume that the difference in composition of the gases from the two fuels under discussion, when the rates of gasification and operating conditions are the same, is due to the difference in the amount of surface exposed to the action of the gases or to the difference in the number of pieces of coal in the path of the gases or to a combination of the two, or, in other words, to the effect of the size of the fuel.

The above conclusion is also supported by operating experience. Shallow fuel beds, which are equivalent to the use of large fuel, or small catalytic surface, produce high CO_2, high temperature of gases and low CO. The tests 18 and 19 show these conditions. The efficiency does not drop off, consequently the temperature of the gas is not high; for the excessive heat liberated, as evinced by the high CO_2, is utilized in the production of hydrogen.

16. *The Effect of Capacity.*—The effect of capacity, or of the rate of gasification upon the efficiency of the producer, when operating with the different fuels, is illustrated by Fig. 24, 25 and 26.

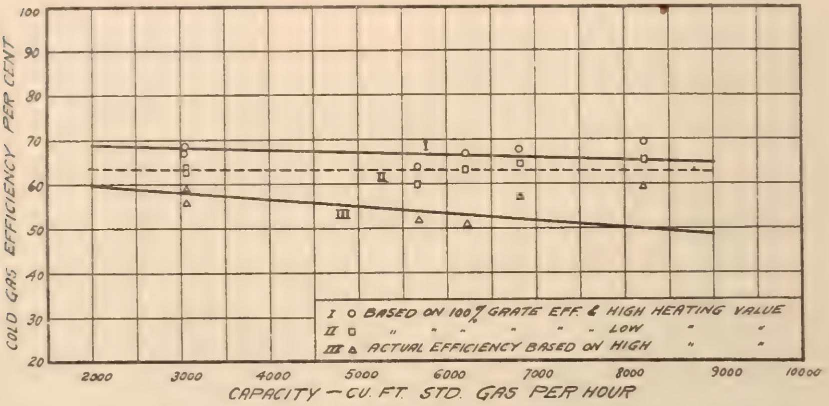

FIG. 24

CAPACITY—EFFICIENCY CURVES FOR PHILADELPHIA AND READING COAL

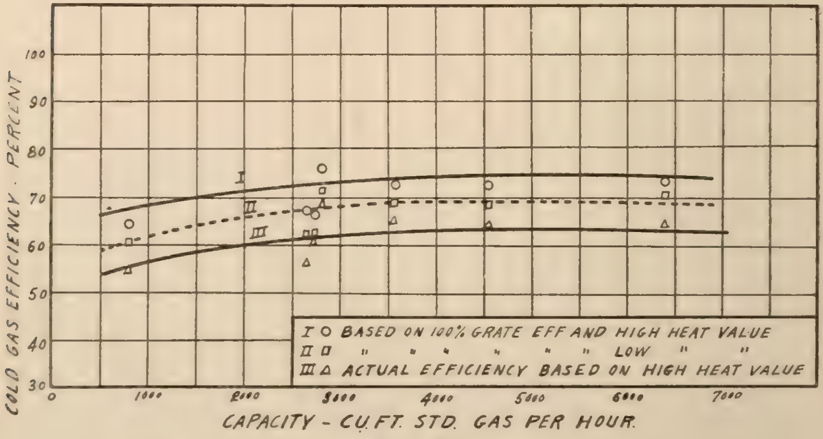

FIG. 25

CAPACITY—EFFICIENCY CURVES FOR LEHIGH VALLEY COAL

As there were only three tests for the gas house coke, no curves have been plotted from these tests. For each of the other fuels, three curves have been plotted, one for the actual efficiency based on high heating value of the gas, one for the efficiency based on 100 per cent grate efficiency, or, which is the same, based on combustible and upon the high heating value of the gas, and a third based on combustible and the low heating value of the gas. This latter curve is shown dotted on the figures.

Fig. 24 shows the curves for the Philadelphia and Reading coal. The curve for the actual efficiency drops off at the higher capacities. This is principally due to the more rapid formation of clinker, as the capacity and the temperature increase, which results in the necessity for more frequent poking, consequently, a much higher grate loss. The curve for efficiency based on 100 per cent grate efficiency and the high heating value of the gas drops off slightly. The curve based on 100 per cent grate efficiency and upon low heating value is practically a straight line.

The curves of Fig. 25 show the results for the Lehigh Valley coal. The drop in the curve for the lower capacities is partially due to the sensible heat lost to the producer walls and fuel bed, on account of the lower temperature of the fuel bed at the start of the test. This has been pointed out in the discussion of the unaccounted-for losses. If a correction were made for this unaccounted for loss, the curves would approach a straight line. The drop at the other end of the curve is largely due to the effect of the velocity of the gases through the fuel bed, or to the shorter time of contact of gases with fuel owing to the larger size of this fuel.

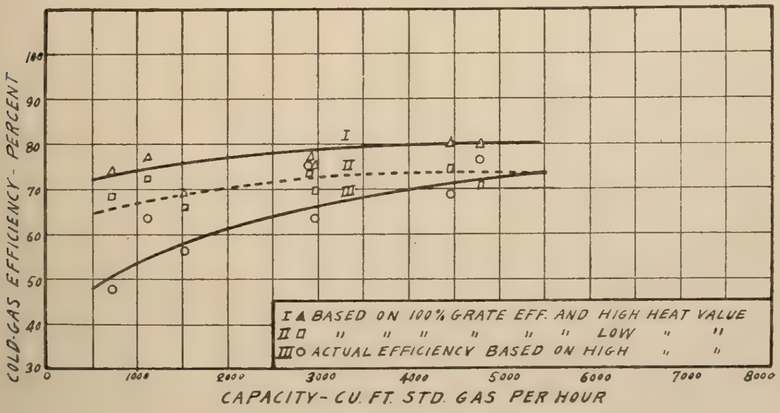

FIG. 26
CAPACITY—EFFICIENCY CURVES FOR SCRANTON PEA COAL

The curves for the Scranton pea coal, Fig. 26, show very little tendency to drop off at the highest capacity. This, it is believed, is explained by the small size of the fuel, consequently, by the larger area of fuel surface exposed to the gases.

The rapid falling off of the actual efficiency curve on these tests is due to the sensible heat lost to the producer walls and fuel bed as previously noted, and also to the fact that in the case of the Lehigh and Scranton coals, which had no tendency to clinker, that the weight of ash and refuse tended to remain constant and was independent of the capacity. As the composition of the ash and refuse remained constant, the actual grate loss tended to remain constant, which would, therefore, cause the actual efficiency at light loads to drop off. The curve based on 100 per cent grate efficiency is practically a straight line slanting toward the low capacities; this slant is also due to the sensible heat lost to the walls and the fuel bed of the producer.

The results of Table 6 bear out the conclusion drawn from the curve as to the effect of capacity. These results together show that capacity, or the rate of gasification, within very wide limits has little effect upon either the efficiency, the composition of the gas, or upon the heating value, so long as the steam supplied to the producer is so regulated as to maintain the same temperature within the fuel bed.

17. *Coal Per Square Foot of Grate Area*—The weight of coal gasified per square foot of grate area is related directly to the rate of gasification and depends upon this. Owing to the general use of the term and to the interest attached thereto by engineers, it probably merits a separate discussion.

It will be seen from Table 5, item 66, that this quantity varies in the tests on the Lehigh and Scranton coals from 7.67 lb. per hour, in the case of test 21, to 49.8 lb. in the case of test 17. Within this range, the producer has been operated with satisfactory results.

Anthracite producers in this country are rated on a basis of from 10 to 15 lb. of coal per sq. ft. of grate area per hour by the manufacturers. In European practice, the rating is from 20 to 30 lb. of coal per square foot of grate area. This difference in rating is doubtless due to the difference in the fuels used in the two countries.

From the results of the present tests, it must be concluded that the coal gasified per square foot of grate area depends almost entirely upon the nature of the coal. Were it not for practical considerations there seems to be no reason why the coal gasified should depend upon any condition other than the depth of the fuel bed. From the works of Dr. Clement previously referred to, and of Boudouard and others, it will be seen that a certain length of time is required for the completion of the reaction within the fuel bed of the producer, also that the time required for their completion depends upon the temperature and upon the catalytic action of the fuel. Consequently, in order to attain high rates of gasification, it is necessary that the fuel bed be increased in depth in order that the proper time of contact of gases with fuel obtains.

In the actual operation of the producer with a given depth of fuel bed, the coal gasified per square foot of grate area depends upon the amount of ash, the nature of the coal, and upon the catalytic action of the fuel.

The larger the percentage of ash in the fuel, the more frequently the producer requires cleaning, so that with high rates of gasification the removal of the ash alone would place a limit. The nature of the ash, that is, the temperature of fusing, also places a limit for a given depth of fuel bed. An ash fusing at a low temperature requires a low temperature within the fuel bed of the producer. This results in requiring a longer time of contact of gases with fuel in order that the reaction within the fuel bed may be completed. A fuel with slow catalytic action produces the same result.

Since, from the above, the coal gasified per square foot of grate area depends upon the depth of the fuel bed, we have used the expression "rate of descent of dry coal through the fuel bed" or " coal burned per cubic foot of fuel bed", to express the performance of a producer in addition to the expression under discussion.

18. *Clinker.*—The formation of clinker is due to the presence of incombustible matter in the ash which fuses at the temperature maintaining in the fuel bed of the producer. There are two ways of dealing with this, the more practical one being the operation at such capacities and with such an amount of water, or CO_2, in the case of the carbon monoxide producer, as to keep the temperature below that of the fusing point of the earthy matter contained

in the fuel. In doing this, the efficiency of the producer will be somewhat lowered, as the reaction of CO_2 on carbon will be greatly lessened, and large quantities of steam must be used to keep the temperature down, through the formation of hydrogen and the heating of the moisture that escapes decomposition. The heating of this moisture results in the lowering of the efficiency of the producer. The curves of Fig. 27, 28 and 29 indicate that the

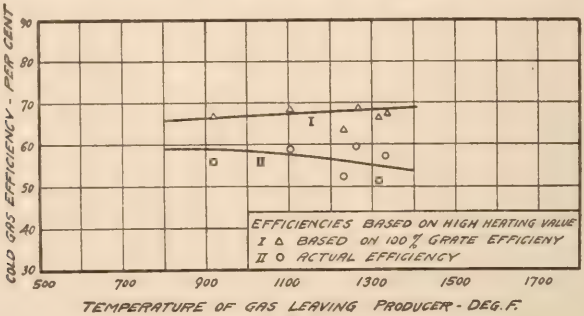

FIG. 27

TEMPERATURE—EFFICIENCY CURVES FOR PHILADELPHIA AND READING COAL

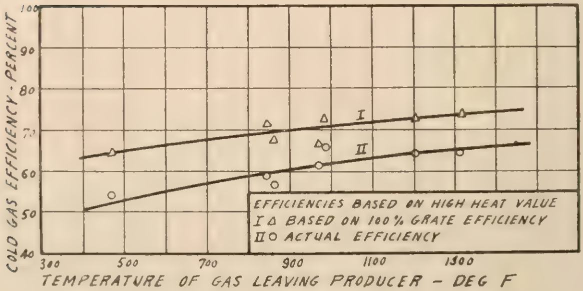

FIG. 28

TEMPERATURE—EFFICIENCY CURVES FOR LEHIGH VALLEY COAL

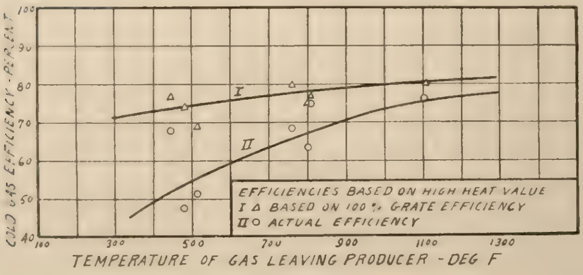

FIG. 29

TEMPERATURE—EFFICIENCY CURVES FOR SCRANTON PEA COAL efficiency of the producer tends to increase with the higher temperatures of operation. This increase, however, is comparatively small within certain limits and since reliability of operation is the desideratum, and not necessarily efficiency, the obvious means of decreasing the irregularities due to the formation of clinker is by operating at lower temperatures.

It is possible that fuels may be found in which the formation of clinker begins at a temperature so low as to prevent the reactions within the producer taking place. This is, however, not probable. The water vapor in practically every case tends to disintegrate both fuel and clinker. If an excess is used, it will tend to cause the fuel bed to become mushy and will result in poor gas, and a shut down if engines are operating on the gas.

A second method of dealing with a coal containing a fusible ash is to add a flux in the form of a limestone or other cheap material which will render the clinker so fusible as to cause it to run from the fuel bed of the producer. Test 14 on the Philadelphia and Reading coal was run in this manner. It was determined from an analysis of the ash that about 60 lb. of limestone ($CaCO_3$) would be required per 100 lb. of coal in order to form a highly fluid clinker. The fuel charged on this test was, therefore, mixed with limestone in the ratio of 100 lb. of coal to 60 lb. of limestone. As the limestone when heated gives out 44 per cent of its weight in CO_2, this CO_2 takes the place of the water vapor and keeps the temperature down through the CO_2, reacting on the C to form CO.

The efficiency of the test is low, due partly to the absorption of heat in driving off the CO_2 from the limestone, but largely to the loss through the difference in temperature of the fuel bed at the start and close of the test. The graphical log sheet, Fig. 30, shows the uniformity of conditions that maintained. The fuel bed required practically no poking during the test, as the clinker fused with the limestone and trickled into the ash pit. At the close of the test, however, when the fire doors were opened for cleaning, the entire fuel bed "froze", and had to be broken up at the expense of considerable time and labor.

The above method is mentioned simply as a matter of interest, and indicates a possible though hardly practical means of dealing with clinker.

19. *Weight of Water Required for the Producer.*—The weight of

water required by the producer depends very largely on the fuel that is being used, and to a certain extent upon the proportions of the producer.

With a fuel that has no tendency to clinker, the highest efficiency will be obtained by using such an amount of water as

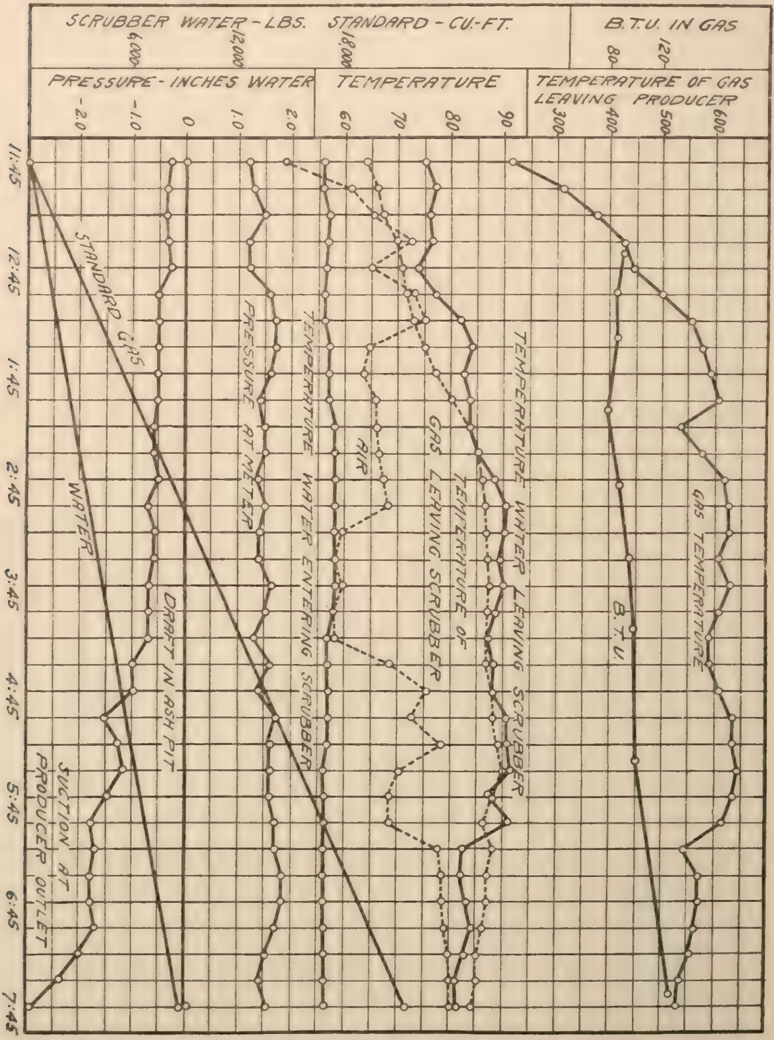

FIG. 30

to result in between 2 and 4 per cent CO_2 in the gas leaving the producer. This of course, requires a deep fuel bed and a high temperature within the decomposition zone to insure the completion of the CO reaction. It is a condition of operation that can scarcely be obtained in a producer as small as the one that has been used on the present tests, for the reason that while there may be no formation of clinker, there will be, owing to the small diameter of the fuel bed, a tendency for the fuel to pack, which will necessitate poking from time to time in order to insure the proper descent of the fuel.

This packing or arching of the fuel, as it decreases the time of contact of gases with fuel, results in increasing the percentage of CO_2, and raising the temperature of the gases, which tends to lower the efficiency. If the producer had been provided with some form of shaking grate, which is quite possible in producers of this size, the efficiency would have been increased several per cent and the uniformity of operation increased much more.

In the case of fuels containing an ash fusing at temperatures close to 2200° F., it will be necessary to use sufficient water to lower the temperature of the fuel bed below that temperature at which the formation of clinker begins. This may result in a slightly lower efficiency of operation, but as it is usual that such operation is necessitated through the use of a poor, consequently a cheap fuel, the efficiency is of less importance than the reliability of operation. In the case of such operation, the hydrogen in the gas will be high, the CO_2 high and the CO low.

Test 23 of Table 6 as compared with Test No. 24 shows the effect of operating with a large quantity of moisture, on the CO_2, heating value and efficiency.

As has been indicated a number of times, the reaction of water on incandescent carbon to form hydrogen takes place much faster at the lower temperature than does the CO reaction, consequently, when a producer is operating with a shallow fuel bed, or with a larger fuel, as in the case of the Lehigh coal, or with a fuel of slow catalytic action, in order to keep down the temperature of the gases and to keep up the efficiency, it will be necessary to use larger quantities of water than would be the case when operating with a deeper fuel bed, as there will be insufficient time for the completion of the CO reaction unless the producer is operated at a greatly lowered capacity.

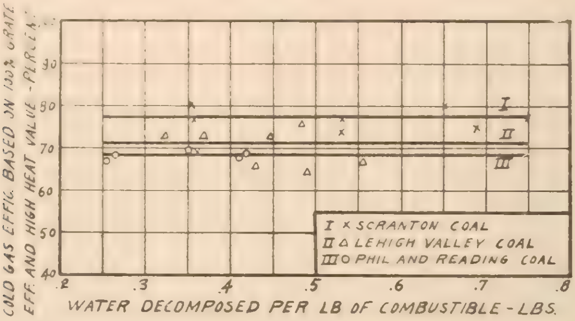

FIG. 31

The curves of Fig. 31 show that within the accuracy and range of these tests, there is practically no effect upon the efficiency of the producer due to the increase of the water decomposed.

If the range were increased in either direction, there would be a great falling off in efficiency.

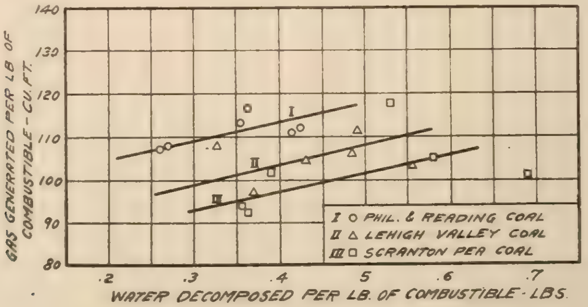

FIG. 32

Fig. 32 shows the effect of the water decomposed per pound of combustible upon the volume of gas generated per pound of combustible, and Fig. 33 shows the effect of the gas generated per pound of combustible upon the heating value of the gas per cu. ft.

20. *The Effect of Fuel Bed Temperature.*—The effect of the fuel bed temperature upon the composition of producer gas and upon the efficiency of the producer from the previous discussion has been pointed out. From the results of Harries' experiments, (see page 6). the experiments of Dr. Clement, illustrated

by Fig. 21, 22, and 23, and from the results of Boudouard's experiments, the general effect of the fuel bed temperature can be predicted with considerable certainty. The exact effect, as it depends upon the catalytic action of the fuel, the size of the fuel, and other variables, can not, however, be determined for the different fuels without direct experiment. Within comparatively wide limits, and, speaking in general, the efficiency of the producer, as has been pointed out, is practically independent of the amount of steam fed to the producer or the amount of H_2 in the gas. With low fuel bed temperatures, it will be seen from Dr. Clement's curves that the CO will be low and the CO_2 high. From the experiments of Harries, it will be seen that the reaction of H_2O on carbon has reached equilibrium at a temperature as low as 1240° F. with 8.41 per cent of H_2 formed. So therefore in the formation of H_2 at low temperatures and the formation of CO at the higher temperatures, the principal effect of changing the temperature of the fuel bed is to cause a shifting in the relative percentage of CO, CO_2, and H_2 present in the gas generated. If the change is so made that there is no great difference in the temperature of the leaving gases, the efficiency of the producer is very slightly affected. Tests 23 and 24 of Table 6 illustrate this. The fuel bed temperature in the case of test 23 is 1740°F., the tem-

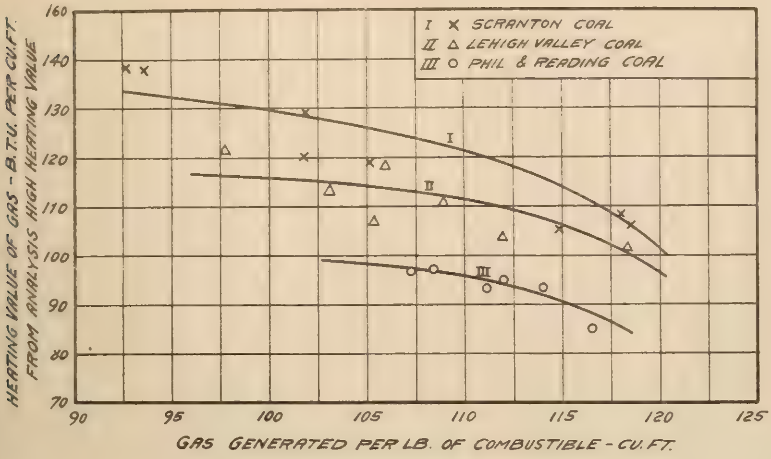

FIG. 33

perature of the fuel bed for test 24 is approximately 2200 F. The temperature of the gases leaving the producer is practically the same, as is also the efficiency. The percentage of CO has shifted from 27.3 to 17.39, while the H_2 has shifted from 10.8 to 16.96 per cent. The heating value of the gas for the lower temperature has fallen off, while the number of cubic feet of gas per pound of combustible has increased from 92.6 for test 24 to 101.41. This latter is due to the lesser density of the hydrogen.

The fuel bed temperatures, as obtained for the different tests, are given in Table 3.

The radiation and conduction losses will probably not exceed 1 per cent. The loss due to tar and soot and to the absorption of the gases by the scrubber water will not exceed 1 per cent. The losses due to the difference between the initial and final temperatures of the fuel bed are variable and depend upon the capacity and to a certain extent upon the length of time required to blow the fires and produce a gas sufficiently rich for operating conditions. These losses have been discussed on page 37, and it was shown here that they would doubtless vary from .5 to about 9 per cent, depending upon the above conditions.

The experimental errors made in the measurement of different quantities may be either positive or negative and the magnitude of the probable error varies from about 2 or 3 per cent in tests at capacities between 4000 and 2800 cubic feet of gas per hour to between 4 and 7 per cent for tests between 2000 and 750 cubic feet per hour.

IV. CONCLUSIONS

1. A producer of the size and type tested is a practical piece of apparatus. With proper care in operating, and in the selection of the fuel, it is also a reliable piece of apparatus possessed of a fair degree of efficiency.

2. The efficiency of the producer is, within wide limits, practically independent of the rate of gasification.

3. The efficiency of the producer, within comparatively wide limits, is only slightly affected by the relative percentages of H_2 and CO appearing in the gas. With low temperature in the fuel bed, due to the use of a large quantity of steam, there is a high percentage of H_2 and CO_2. With high temperature in the fuel bed, the H_2 and CO_2 drop off while the CO increases.

4. From the above, it follows that the amount of steam to be used under any given conditions depends entirely upon the nature of the fuel. Clinkering coals require large quantities of steam in order to keep the temperature below that of the formation of clinker, while non-clinkering coals require much less steam.

5. The size of the fuel for a given temperature of fuel bed and a given depth of fuel bed has a marked effect upon the heating value and composition of the gas, and upon the efficiency of the producer. An increase in the size of fuel tends to cause the efficiency, heating value of the gas, and the percentage of CO to drop off.

6. The graphical logs show that the conditions (with the exception of the tests on Philadelphia and Reading coal) were sufficiently uniform to prevent trouble in the operation of engines driving machinery that requires no great degree of sensitiveness in regulation, such as the driving of pumping machinery, or shafting for machine shops, provided these engines have overload capacities of from 15 to 20 per cent. The conditions were not sufficiently uniform to permit the use of the gas in engines designed to drive electric lighting and other machinery requiring sensitive regulation. This is due, however, to the very small size of the unit. With larger units, the uniformity has proved entirely sufficient for the operation of engines driving electric generators.

7. From item 111 of Table 5, it will be found that the volume of gas generated per lb. of coal averages about 76 cu. ft. The average low heating value of the gas, if the tests on the Philadelphia and Reading coal are not used, is approximately 110 B. t. u. per cu. ft. Since the average engine requires about 10,500 effective B. t. u. per b. h. p. hr., 95 cu. ft. of this gas will be required per b. h. p. hr. or $\frac{95}{76} = 1.25 +$ lb. coal. Since the coal contains about 12,800 B. t. u. per lb., the efficiency of the plant will be $\frac{2545 \times 100}{12800 \times 1.25} = 16$ per cent. approximately. The thermal efficiency of a steam plant of this same size will be about 3 per cent, assuming a boiler efficiency of 60 per cent and an engine efficiency of 5 per cent.

If the anthracite fuel costs $5.00 per ton, soft coal for the steam plant must cost $0.94 per ton in order that the fuel cost remain the same per h. p. hr. The cost of attendance, maintenance and repair will be practically the same for each plant.

8. Under the above conditions, with anthracite at $5.00 per ton, one b. h. p. could be produced at a fuel cost of .31 cent per hr., or 3.7 cents per 12 hrs.

9. The effect of the capacity, the amount of steam used, the effect of the size of the fuel, the amount of CO_2 in the gas, and other items of a general nature, and the conclusions drawn from such items,are entirely applicable to producers of all sizes. It is believed that these items, with the forms and formulas, constitute the most valuable portions of this paper.

APPENDIX

APPENDIX *

FORM 1 RESULTS OF GAS PRODUCER TRIALS

1 Test number..
2 Made by..
3 At ....The University of Illinois..........
4 Kind of producer ....Otto ....
5 To determine ....Efficiency..........................
6 Principal conditions governing trial....Uniform load..........................
7 Kind of fuel..........................Scranton-Anthracite..........................
8 Kind of grate..........................Plain ....
9 Method of starting and stopping test....Alternate..........................
10 Type of producer..........................Suction ....
11 Form of blower-ejector..........................Schutte & Koerting..........................
12 Date of trial..
13 Duration of trial..

DIMENSIONS AND PROPORTIONS

14 Dimensions of grate, ft..
15 Grate area, sq. ft..
16 Mean diameter of fuel bed, ft..
17 Depth of fuel bed, ft..
18 Area of fuel bed, sq. ft..
19 Height of discharge pipe above grate, ft..
20 Approximate width of air spaces in grate, inches..
21 Area of air space, sq. ft..
22 Proportion of air space to whole grate area, per cent..
23 Area of discharge pipe, sq. ft..
24 Water heating surface in vaporizer, sq. ft..
25 Outside diameter of shell, ft..
26 Length of shell from base to top of magazine, ft..
27 Ratio of water heating surface to grate area,—to 1..
28 Ratio of minimum draft area to grate area, 1 to..

AVERAGE PRESSURES

29 Draft in ashpit, inches, water..
30 Suction at producer outlet, inches, water..
31 Pressure at meters, inches, water..
32 Corrected barometer reading..
32.1 Steam pressure, lb. per sq. in. gage..

AVERAGE TEMPERATURES

33 Of fire room, deg. Fahr..
34 Of steam leaving vaporizer, deg. Fahr..
35 Of feed water entering vaporizer, deg. Fahr..
36 Of overflow from vaporizer, deg. Fahr..
37 Of water entering scrubber, deg. Fahr..
38 Of water leaving scrubber, deg. Fahr..
39 Of gases leaving producer, deg. Fahr..
40 Of gases leaving scrubber, deg. Fahr..
41 Of gases entering meter, deg. Fahr..

FUEL

42 Size and condition..
43 Weight of coal as fired, lb..
44 Percentage of moisture in coal..
45 Total weight of dry coal fired, lb..

---

*In the following Appendix, the forms used in computing the results are given for the benefit of those who may have occasion to use them in the working of results of producer tests. The use of the forms has been explained on page 38.

46 Total ash and refuse, lb. ....
47 Quality of ash and refuse..
48 Total combustible consumed, lb..
49 Percentage of ash and refuse in dry coal ....

PROXIMATE ANALYSIS OF COAL

50 Fixed carbon ....
51 Volatile matter..
52 Moisture ....
53 Ash ....
54 Sulphur separately determined..

ULTIMATE ANALYSIS OF DRY COAL

55 Carbon, C..
56 Hydrogen, H_2..
57 Oxygen, O_2..
58 Nitrogen, N_2..
9 Sulphur, S..
60 Ash ....
61 Moisture in sample of coal as received..

ANALYSIS OF DRY ASH AND REFUSE

62 Carbon, per cent..
63 Earthy matter, per cent..
  a SiO_2..
  b { Al_2O_3..
     Fe_2O_3..
  c MgO..
  d CaO..

FUEL PER HOUR

64 Dry coal fired per hr. lb..
65 Combustible consumed per hr. lb..
66 Dry coal per sq. ft. of grate area per hr. lb..
67 Combustible per sq. ft. of grate area per hr. lb..
68 Dry coal per sq. ft. of fuel bed per hr. lb..
69 Combustible per sq. ft. of fuel bed per hr. lb..
70 Rate of descent of dry coal through fuel bed, lb per ft. per sq. ft. per hr .....
71 Rate of descent of combustible through fuel bed, lb. per ft. per sq. ft. per hr.

CALORIFIC VALUE OF FUEL

72 Calorific value by oxygen calorimeter per lb. dry coal, B. t. u..
73 Calorific value by oxygen calorimeter per lb. of combustible B. t. u..
74 Calorific value by analysis per lb. dry coal. B. t. u..
75 Calorific value by analysis per lb. of combustible, B. t. u ....

WATER

76 Total[1] weight of water fed to vaporizer, lb..
77 Total weight of overflow from vaporizer, lb..
78 Water[1] actually evaporated in vaporizer, lb..
79 Total weight of water fed to producer, lb..
  a From vaporizer[1]..
  b In air..
  c In coal..
80 Total weight of water decomposed. ....
81 Total weight of water in gas leaving producer, lb..
82 Ratio of water decomposed to water supplied..
83 Weight of water decomposed per lb. gas generated, lb..

---

[1] Steam fed to producer, where vaporizer is not used.

84 Weight of water decomposed per lb. of dry coal fired, lb.
85 Weight of water decomposed per lb. of combustible consumed, lb.
86 Weight of water decomposed per lb. of air supplied, lb.
87 Weight of water supplied per lb. of dry coal fired, lb.
88 Weight of water per lb. of combustible consumed, lb.
89 Weight of water supplied per lb. of dry air used, lb.
90 Total weight of scrubber water, lb.

WATER PER HOUR

91 Water evaporated per hr. in vaporizer, lb.
92 Water evaporated per hr. per sq. ft. of water heating surface in vaporizer, lb.
93 Weight of water decomposed per hr., lb.
94 Total weight of water fed to producer per hr., lb.
95 Weight of scrubber water used per hr., lb.

QUANTITY OF AIR

96 Per cent of moisture in air, per cent of dry air
97 Total weight of dry air, lb.
98 Total weight of dry air per hr. lb.
99 Weight of dry air used per lb. of dry coal fired, lb.
100 Weight of dry air used per lb. of combustible consumed, lb.
101 Weight of dry air used per lb. of dry gas generated, lb.

GAS

102 Per cent moisture in gas leaving producer, per cent of dry gas
103 Per cent of soot and tar in gas leaving producer
104 Calorific value of standard gas from analysis (high value) B. t. u. per cu. ft.
104*a* Calorific value of standard gas from analysis (low value) B. t. u.
105 Calorific value of standard gas from calorimeter, (high value) B.t.u. per cu. ft.
106 Specific weight of standard gas, lb. per cu. ft.
107 Specific heat of dry gas leaving producer
108 Carbon ratio C/H
109 Total volume standard gas, cu. ft.
110 Volume of standard gas per hr. cu. ft.
111 Volume of standard gas per lb. of dry coal.
112 Volume of standard gas per lb. of combustible.
113 Total weight of standard gas, lb.
114 Weight of standard gas per hr., lb.
115 Weight of standard gas per lb. of dry coal fired, lb.
116 Weight of standard gas per lb. of combustible consumed, lb.

GAS ANALYSIS BY VOLUME

117 Carbon dioxide, CO_2
118 Carbon monoxide, CO
119 Oxygen, O_2
120 Hydrogen, H_2
121 Marsh gas, CH_4
122 Olefiant gas, C_2H_4
123 Sulphur dioxide, SO_2
124 Hydrogen sulphide, H_2S
125 Nitrogen, N_2, by difference

EFFICIENCY

126 Grate efficiency, per cent.
127 Hot gas efficiency, based on high heating value, per cent.
128 Cold gas efficiency, based on high heating value, per cent.
128*a* Cold gas efficiency, based on low heating value, per cent

EFFICIENCY BASED ON 100 PER CENT GRATE EFFICIENCY

128*b* Hot gas efficiency, based on high heating value, per cent..................
128*c* Cold gas efficiency, based on high heating value, per cent..................
128*d* Cold gas efficiency, based on low heating value, per cent ....

COST OF GASIFICATION

129 Cost of fuel per ton delivered in producer room..........................
130 Cost per 1000 cu. ft. of standard gas, cents..........................
131 Cu. ft. scrubber water per 1000 cu. ft. gas..........................

POKING

132 Method of poking..........................
133 Frequency of poking..........................

FIRING

134 Method of firing..........................
135 Average intervals between firing..........................
136 Average amount of fuel charged each time, lb..........................

HEAT BALANCE

DEBIT B. T. U.

a Total heat supplied per lb. dry coal............
b Total heat supplied by air per lb. dry coal......
c Total heat supplied by moisture in air per lb. dry coal..................
d Total heat supplied by moisture in coal per lb. dry coal..................
e Total heat supplied as sensible heat in coal per lb. dry coal..................
f Total[1] heat supplied by water in vaporizer per lb. dry coal..................

Total..................

CREDIT B. T. U. Per Cent

a Heat contained as sensible heat in dry gas.....
b Heat contained in moisture..................
c Heat contained in dry gas (heat of combustion)
d Heat in unburned carbon..................
e Heat contained in ash and refuse as sensible heat
f Heat lost in overflow from vaporizer..........
g Heat lost in radiation and conduction..........

Total..................

FORM 2 RESULTS OF GAS PRODUCER TRIALS

NO. OF TEST DATE, TIME OF START,
TIME OF STOP DURATION OF TRIAL, HRS.
KIND OF FUEL.

DIMENSIONS AND PROPORTIONS

1 Dimensions of grate, ft..........................
2 Grate area sq. ft..........................

---

[1] Supplied in steam, where vaporizer is not used.

3 Mean diameter of fuel bed, ft..........
4 Depth of fuel bed, ft..........
5 Area of fuel bed, sq. ft..........
6 Height of discharge pipe above grate, ft..........
7 Approximate width of air spaces in grate, inches..........
8 Area of air spaces sq. ft..........
9 Ratio of air space to whole grate area, %..........
10 Area of discharge pipe, sq. ft..........
11 Water heating surface in vaporizer, sq. ft..........
12 Outside diameter of shell, ft..........
13 Length of shell from base to top of magazine, ft..........
14 Ratio of water heating surface to grate area —to 1..........
15 Ratio of minimum draft area to grate area to..........

AVERAGE PRESSURES

16 Average barometer reading, inches Hg..........
17 Average corrected barometer reading, inches Hg..........
18 Draft in ash pit, inches water..........
19 Suction at producer outlet, inches water..........
20 Absolute pressure at producer outlet, inches Hg..........
21 Suction[1] at orifice, inches water..........
22 Absolute pressure[1] at orifice, inches Hg..........
23 Pressure at meters, inches water..........
24 Absolute pressure at meters, inches Hg..........
25 Vapor pressure at meters, inches Hg..........
26 Dry gas pressure at meters, inches Hg..........
27 Suction at meter for dryer, inches water..........
28 Absolute pressure at meter for dryer, inches Hg..........

AVERAGE TEMPERATURES

29 At barometer, deg. Fahr..........
30 Of fire room, deg. Fahr..........
31 Of fire room, deg. absolute Fahr..........
32 Of steam, deg. Fahr..........
33 Of feed water entering vaporizer, deg. Fahr..........
34 Overflow from vaporizer, deg. Fahr..........
35 Rise in vaporizer, deg. Fahr..........
36 Of water entering scrubber, deg. Fahr..........
37 Of water leaving scrubber, deg. Fahr..........
38 Rise in scrubber, deg. Fahr..........
39 Of gases leaving producer, deg. Fahr..........
40 Of gases leaving producer, deg. abs. Fahr..........
41 Of gases leaving first scrubber, deg. Fahr..........
42 Of gases leaving first scrubber, deg. abs. Fahr..........
43 Drop in temperature of gases in scrubber, deg. Fahr..........
44 Of gases entering meters, deg. Fahr..........
45 Of gases entering meters, deg. abs. Fahr..........
46 Of gas at meter at dryer, deg. Fahr..........
47 Of gas at meter at dryer, deg. abs. Fahr..........

FUEL

48 Size and condition..........
49 Weight of coal as fired, lb..........
50 Percentage of moisture in coal..........
51 Total weight of dry coal fired, lb..........

---

[1] Steam pressure may be substituted here in case the water is not supplied from the vaporizer

52 Total ash and refuse, lb
53 Quality of ash and refuse
54 Total weight of combustible, lb
55 Percentage of ash and refuse in dry coal, per cent

PROXIMATE ANALYSIS OF COAL

56 Fixed carbon, per cent
57 Volatile matter, per cent
58 Moisture, per cent
59 Ash, per cent
60 Sulphur, separately determined, per cent

ULTIMATE ANALYSIS OF DRY COAL

61 Carbon, C, per cent
62 Hydrogen, H_2, per cent
63 Oxygen, O_2, per cent
64 Nitrogen, N_2, per cent
65 Sulphur, S, per cent
66 Ash, per cent
67 Moisture in sample coal as received, per cent

ANALYSIS OF DRY ASH AND REFUSE

68 Carbon, per cent
69 Earthy matter, per cent
- *a* SiO_2
- *b* Al_2O_3 Fe_2O_3
- *c* MgO
- *d* CaO

FUEL PER HOUR

70 Dry coal fired per hr., lb
71 Combustible consumed per hr., lb
72 Dry coal per sq. ft. of grate area per hr., lb
73 Combustible per sq. ft. of grate area per hr., lb
74 Dry coal per sq. ft. of fuel bed per hr., lb
75 Combustible per sq. ft. of fuel bed hr., lb
76 Rate of descent of dry coal through fuel bed, lb. per ft, per sq. ft. per hr.
77 Rate of descent of combustible through fuel bed, lb. per ft. per sq. ft. per hr

CALORIFIC VALUE OF FUEL

78 Calorific value by oxygen calorimeter per lb. dry coal, B. t. u
79 Calorific value by oxygen calorimeter per lb. combustible B. t. u
80 Calorific value by analysis, per lb. dry coal, B. t. u
81 Calorific value by analysis, per lb. combustible, B. t. u

WATER

82 Total[1] weight fed to vaporizer, lb
83 Total weight of overflow, lb
84 Water[1] actually evaporated in vaporizer, lb
85 Weight of water fed to producer,
- *a* From vaporizer[1]
- *b* In air
- *c* In coal
- Total

86 Total weight of water decomposed from analysis, lb
87 Total weight of water decomposed as used in calculations, lb

---

[1] Steam fed to producer where vaporizer is not used.

88 Total weight of moisture in gas leaving producer, lb.
89 Ratio of water decomposed to water supplied
90 Weight of water decomposed per lb. of gas generated, lb.
91 Weight of water decomposed per lb of dry coal fired, lb
92 Weight of water decomposed per lb. of combustible consumed, lb.
93 Weight of water decomposed per lb. of air supplied, lb.
94 Weight of water supplied per lb. of dry coal fired lb
95 Weight of water supplied per lb. of combustible consumed, lb
96 Weight of water supplied per lb. of air used, lb
97 Total weight of scrubber water, lb
98 Total weight of water absorbed from sample by dryer, grams

WATER PER HOUR

99 Water evaporated per hr. in vaporizer, lb
100 Water evaporated per hr. per sq. ft. of water heating surface in vaporizer, lb
101 Weight of water decomposed per hr., lb
102 Total weight of water fed to producer per hr., lb
103 Weight of scrubber water used per hr., lb

QUANTITY OF AIR

104 Relative humidity of air, per cent
105 Per cent of moisture contained in air, per cent by weight of dry air
106 Total weight of dry air by analysis, lb
107 Total weight of dry air by orifice, lb
108 Total weight of dry air as used in calculations, lb
109 Weight of dry air per hr. from total used in calculations
110 Weight of dry air used per lb. of dry coal fired, lb
111 Weight of dry air used per lb. of combustible consumed, lb
112 Weight of dry air used per lb. of dry gas generated, lb

GAS

113 Volume of gas sample passing through meter at dryer, cu. ft
114 Volume of standard gas passing through meter at dryer, cu. ft
115 Total weight of gas passing through dryer meter, lb
116 Percentage of moisture in gas leaving producer, from dryer, per cent dry gas
117 Percentage of moisture in gas leaving producer, from water fed to producer, per cent dry gas
118 Percentage soot and tar in gas leaving producer, per cent
119 Calorific value per cu. ft. standard gas from analysis B.t.u. (high value)
119*a* Calorific value per cu. ft. of standard gas, by analysis (low value)
120 Calorific value per cu. ft. of standard gas from calorimeter, B.t.u. (high value)
121 Specific weight of standard gas, lb. per cu. ft
122 Specific heat of dry gas leaving producer
123 Carbon ratio C/H
124 Total volume of gas from meters, cu. ft
125 Total volume of standard gas, from meters, cu. ft
126 Total volume of standard gas, from analysis, cu. ft
127 Total volume as used in calculations, cu. ft
128 Volume of standard gas per hr. from total used in calculations
129 Volume of standard gas per lb. of dry coal from total used in calculations, cu. ft.
130 Volume of standard gas per lb. of combustible from total used in calculations, cu. ft
131 Total weight of standard gas from total used in calculations, lb
132 Weight of standard gas per hr., lb
133 Weight of standard gas per lb. of dry coal, lb
134 Weight of standard gas per lb. of combustible, lb

GAS ANALYSIS BY VOLUME

135 Carbon dioxide CO_2 ....
136 Carbon monoxide, CO ....
137 Oxygen, O_2 ....
138 Hydrogen, H_2 ....
139 Marsh gas, CH_4 ....
140 Olefiant gas, C_2H_4 ....
141 Sulphur dioxide, SO_2 ....
142 Hydrogen sulphide, H_2S ....
143 Nitrogen, N_2 by difference ....

GAS ANALYSIS BY WEIGHT

144 Carbon dioxide, CO_2 ....
145 Carbon monoxide, CO ....
146 Oxygen, O_2 ....
147 Hydrogen, H_2 ....
148 Marsh gas, CH_4 ....
149 Olefiant gas, C_2H_4 ....
150 Sulphur dioxide, SO_2 ....
151 Hydrogen sulphide, H_2S ....
152 Nitrogen, N_2 by difference ....

EFFICIENCY

153 Grate efficiency, per cent ....
154 Hot gas efficiency, based on high heating value, per cent ....
155 Cold gas efficiency, based on high heating value, per cent ....
155.1 Cold gas efficiency, based on low heating value, per cent ....

EFFICIENCY BASED ON 100 PER CENT GRATE EFFICIENCY

155*a* Hot gas efficiency, based on high heating value, per cent ....
155*b* Cold gas efficiency, based on high heating value, per cent ....
155*c* Cold gas efficiency, based on low heating value, per cent ....

COST OF GASIFICATION

156 Cost of fuel per ton delivered in producer room ....
157 Cost per 1000 cu. ft. of standard gas, cents ....
158 Cu. ft. scrubber water per 1000 cu. ft. standard gas ....

POKING

159 Method of poking ....
160 Frequency of poking ....

FIRING

161 Method of firing ....
162 Average intervals between firings ....
163 Average amount of fuel charged each time ....

HEAT BALANCE

DEBIT B.T.U.

a Total heat supplied per lb. dry coal ....
b Total heat supplied by air per lb. dry coal ....
c Total heat supplied by moisture in air per lb. dry coal ....
d Total heat supplied by moisture in coal ....
e Total heat supplied as sensible heat in coal ....
f Total[1] heat supplied in vaporizer water ....

Total ....

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[1] Supplied in steam, where vaporizer is not used.

CREDIT	B.T.U.	PER CENT
a Heat contained as sensible heat in dry gas..		
b Heat contained in moisture...		
c Heat contained in dry gas (heat of combustion).................		
d Heat in unburned carbon..............		
e Heat contained as sensible heat in ash and refuse...............		
f Heat lost in overflow from vaporizer....		
g Radiation and conduction, by difference......................		
Total.................................		

FORM 3 GUIDE SHEET CONTAINING ALL FORMULAS AND THEIR DERIVATION

The item numbers refer to the items of Form 2, and are arranged in the order of computation.

Item 4. "Depth of fuel bed" is to a certain extent arbitrary. In order that the term may have a fixed and definite meaning we will define it as the distance between the upper edge of the ash zone and that section of the fuel bed from which the gases separate and leave the fuel. The upper edge of the ash zone can ordinarily be readily determined by inspection.

Item 16. This reading is the average of the barometer readings for the test and is not corrected.

Item 17. Item 16 corrected. The following formula may be used:

Let H = corrected barometer reading.

t = temperature, deg. fahr.

h = barometer reading corresponding to temperature t.

Then $H = h\ (1.00254 - 0.000079t)$

Item 17. = Item 16 (1.00254 − 0.000079 × Item 29)

Item 18. = Observed.

Item 19. = Observed.

Item 20. = Item 17 — Item 19 × 0.0735

Item 21. = Observed.

Item 22 = Item 17 — Item 21 × 0.0735

Item 23. = Observed.

Item 24. = Item 17 + Item 23 × 0.0735

Item 25. = Taken from steam tables using temperature in Item 44, 1 lb. per sq. in. = 2.04 in. Hg.

Item 26. = Item 24 — Item 25

Item 27. = Observed.

Item 28. = Item 17 — Item 27 × 0.0735

Items 29 to 47 inc. The observed temperatures should be corrected from the calibration curves before being placed in Form 2. The absolute temperature = the observed temperature + 460 deg.

Item 39. This item is observed in deg. Cent. and should be transferred into deg. Fahr.

$$\text{Deg. Fahr.} = \frac{9}{5}\,\text{deg. Cent.} + 32$$

Each observation must be transferred.

Item 50. Taken from Item 67.

Item 51. Item 49 $\left(1 - \frac{\text{Item 50}}{100}\right)$

Item 52. Taken from ash sheet, correction being made for any moisture taken up in the ash-pit.

Item 54. In these tests the total weight of combustible consumed will be taken as the total weight of dry coal fired, minus
the weight of ash computed from the analysis, minus
the weight of nitrogen — ⅜ × the weight of oxygen, minus
the weight of carbon contained in the ash and refuse and equals

$$\text{Item } 51 - \frac{\text{Item } 51 \times \text{Item } 66}{100} - \frac{\text{Item } 51 \times \text{Item } 64}{100} - \frac{\frac{3}{8}\text{ Item } 51 \times \text{Item } 63}{100}$$

$$-\frac{\text{Item } 52 \times \text{Item } 68}{100}$$

Therefore,

$$\text{Item } 54 = \text{Item } 51 \left[1 - \frac{\text{Item } 66 + \text{Item } 64 + \frac{3}{8}\text{ Item } 63}{100} \right] - \frac{\text{Item } 52 \times \text{Item } 68}{100}$$

Item 55. $= \dfrac{\text{Item } 52 \times 100}{\text{Item } 51}$

Items 56 to 69. From chemist.

Items 69, *a, b, c, d.* The ultimate analysis of the ash will be made only in special cases to obtain data on the formation of clinker.

Item 70. $= \dfrac{\text{Item } 51}{\text{hours}}$

Item 71. $= \dfrac{\text{Item } 54}{\text{hours}}$

Item 72. $= \dfrac{\text{Item } 70}{\text{Item } 2}$

Item 73. $= \dfrac{\text{Item } 71}{\text{Item } 2}$

Item 74. $= \dfrac{\text{Item } 70}{\text{Item } 5}$

Item 75. $\dfrac{\text{Item } 71}{\text{Item } 5}$

Item 76. "The rate of descent of dry coal through the fuel bed," or "The dry coal per cu. ft. of fuel bed per hour," which is the same, offers a means of comparing the rate of gasification in different producers that seems to be better adapted for the purpose than the expressions taken from boiler practice, viz: "coal per sq. ft. of grate area," or "coal per sq. ft. of fuel bed," the latter having been used in producer practice.

Item 76. $= \dfrac{\text{Item } 74}{\text{Item } 4}$

Item 77. $= \dfrac{\text{Item } 75}{\text{Item } 4}$

Item 78. Taken from chemist's report.

Item 79. $= \dfrac{\text{Item } 78 \times (\text{Item } 51 - \text{Item } 52 \times \text{Item } 68) \times 145.40}{\text{Item } 54}$

Item 80. $= \left\{ \text{Item } 61 \times 145.40 + \text{Item } 65 \times 40.00 + [\text{Item } 62 - \tfrac{1}{8} \text{ of Item } 63] \times 620.00 \right\}$

Item 81. $= \dfrac{\text{Item } 80 \times (\text{Item } 51 - \text{Item } 52 \times \text{Item } 68 \times 145.40)}{\text{Item } 54}$

Item 113. Total volume of gas passing through meter at dryer. Observed.

Item 114 Total volume of standard gas passing through meter at dryer. Neglecting the effect of moisture.

Let p_1 = absolute pressure in inches Hg. at dryer meter.
t_1 = absolute temperature, deg. fahr. at dryer meter.
v_1 = total volume of gas passing through meter.
P, V, and T, be the condition of standard gas.
$P = 30$ in. Hg.
$T = 460 + 62 = 522$

Then

$$\frac{p_1v_1}{t_1} = \frac{PV}{T}$$

$$\text{or } V = \frac{p_1v_1T}{Pt_1} = \frac{p_1v_1 \times 522}{30t_1} = \frac{17.4\ p_1v_1}{t_1}$$

from which the value of Item 114 follows.

Item 114. $= 17.4 \dfrac{\text{Item 28} \times \text{Item 113}}{\text{Item 47}}$

Item 118. Not considered in these tests.

Item 119 and 119*a*. One cubic foot of standard gas, i. e. gas at a temperature of 62° Fahr. or 522° absolute and a pressure of 30 in. Hg. gives up the high value in the following table when the products of combustion are brought back to this temperature, and the moisture condensed. If the moisture is not condensed it gives the low value.

H_2 = 328 (high) or 276 (low) B. t. u per cu. ft. of standard gas.
C_2H_4 = 1610 (high) or 1510 (low) B.t.u. per cu. ft. of standard gas.
CO = 319 (both high and low) B. t. u. per cu. ft. of standard gas.
CH_4 = 1010 (high) or 910 (low) B.t.u. per cu. ft of standard gas.

Item 120. This quantity is the average of all the calorimeter determinations. Each separate determination by the calorimeter must be computed and the heating value obtained. The following formula may be used. The calorimeter readings are taken in centigrade units with the exception of the meter reading and pressure.

Let t_2 = temperature of entering water, deg. cent.
t_1 = temperature leaving water, deg. cent.
r = rise in temperature of water, deg. cent.
W = weight of water used during the intervals = 8 litres for all tests.
G_1 = cu. ft. of gas used from meter
t_g = temperature of entering gas, deg. cent.
p_g = pressure entering gas inches Hg. absolute, corrected for vapor pressure of water (see Item 25).
H = heating value per cu. ft. of standard gas (62 deg. Fahr. or 16.7 deg. Cent. and 30 in. Hg.)
t_s = temperature of standard gas = 62 deg. Fahr. or 16.7 deg. Cent.
p_s = pressure of standard gas = 30. in Hg.
G_s = cu. ft. of standard gas.

$$\frac{G_1p_g}{T_g} = \frac{G_sp_s}{T_s}.$$

$$G_s = \frac{G_1 \times p_g \times T_s}{T_g \times p_s}$$

Where T_g and T_s are in absolute deg. cent.,
$t_1 - t_2 = r$
Total heat per cu. ft. standard gas in B.t.u. = H
Total heat absorbed by water = W × r

$$H = \frac{W \times r \times 3.968}{Gs} = \frac{W \times r \times 3.968}{\frac{G1 \times pg \times Ts}{Tg \times ps}}$$

$$= \frac{8 \times r \times 3.968 \times Tg \times 30}{G1 \times pg \times (16.7 + 273)}$$

$$= \frac{Tg \times r \times 3.29}{G1 \times pg}$$

where 3.968 is the conversion factor.

In this formula it is assumed that the exhaust products are brought back to 62° F. This is not strictly true but the error introduced is negligible, when the error in the use of the apparatus is considered. There is another error due to the exhaust products carrying out more or less vapor of water than was brought in by the entering gas and air.

This error will also be small and may either be positive or negative depending on conditions. The entering gas will in most cases come from direct contact with water and will therefore be saturated. The air ordinarily will not be saturated. On combustion, moisture will be formed by the union of the oxygen and hydrogen, there will be a contraction in volume of the gases due to the combustion, and also a contraction or expansion due to a change in temperature after combustion. In whichever direction the change in the weight of moisture in the out-going gas from that brought in by the entering gas may occur, this change may be considered very small; for the contraction on combustion will be comparatively small, and this contraction will partly offset the unsaturated condition of the air used for combustion. Also the change in temperature of the out-going gas from that of the entering gas will be small.

The heating values as given in Items 119 and 120 are the *high values*.

The values obtained from the analysis will be more accurate and will be used in all computations.

Item 121. The specific weights of the following gases at 62 deg. and 30 in. Hg. are

$CO_2 = 0.11610$

$CO = 0.07362$

$O_2 = 0.08418$

$H_2 = 0.00530$

$CH_4 = 0.04278$

$C_2H_4 = 0.07370$

$SO_2 = 0.16380$

$H_2S = 0.08682$

$N_2 = 0.07400$

Item 121. = [Item 135 × 0.1161 + Item 136 × 0.07362 + Item 137 × 0.08418 + Item 138 × 0.00530 + Item 139 × 0.04278 + Item 140 × 0.0737 + Item 141 × 0.1638 + Item 142 × 0.08682 + Item 143 × 0.0740] $\frac{1}{100}$

Item 144 to 152. Calculation of the gas analysis by weight from the analysis by volume. Assume that we have one cubic foot of gas at 62 deg. Fahr. and 20 in. Hg. of the following composition:

VOLUMETRIC ANALYSIS	SPECIFIC WEIGHT	ANALYSIS BY WEIGHT PER CENT
CO_2 = a per cent	$0.1161 = Wa$	$A = \frac{Wa \times a}{W}$
$CO = b$	$0.07362 = Wb$	$B = \frac{Wb \times b}{W}$
$O_2 = c$	$0.08418 = Wc$	$C = \frac{Wc \times c}{W}$
$H_2 = d$	$0.00530 = Wd$	$D = \frac{Wd \times d}{W}$

$CH_4 = e$ $\quad 0.04278 = W_e \quad E = \frac{W_e \times e}{W}$

$C_2H_4 = f$ $\quad 0.07370 = W_f \quad F = \frac{W_f \times f}{W}$

$SO_2 = g$ $\quad 0.16380 = W_g \quad G = \frac{W_g \times g}{W}$

$H_2S = h$ $\quad 0.08682 = W_h \quad H = \frac{W_h \times h}{W}$

$N_2 = i$ $\quad 0.07400 = W_i \quad I = \frac{W_i \times i}{W}$

Where $W = [a \times W_a - b \times W_b - c \times W_c - e \times W_e \ldots\ldots + i \times W_i]\frac{1}{100}$ = Item 121.

Item 122. The specific heats of the gases vary according to the pressure and temperature As the pressure used throughout the experiments is atmospheric we have only to consider the variation with the temperature. The following formulae taken from Zeuner, vol. I, page 147, give the specific heat for constant volume C_v.

$^1CO_2$, mC_v $= 6.50 - 0.00774t$ .... (1)
H_2O, mC_v $= 5.78 - 0.00572t$ .... (2)
O_2 H_2 N_2, CO, mC_v $= 4.76 - 0.00244t$ .... (3)
$mC_p - mC_v$ $= 1.9934$ .... (4)

For the specific heat of marsh gas CH_4, our other constituent, we will use the value $C_p = 0.6$. This is approximate, but as the quantity of CH_4 is small the resultant error is consequently small.

In the above formula, m is the molecular weight of the gas, t the temperature in deg. cent. and C_v the mean specific heat between zero and t deg. cent. C_p is determined from formula (4). From the above formulas, the analysis by weight as determined below and the temperature of the gases leaving the producer, the specific heat of each constituent in a unit weight of the gas may be determined. The specific heat of the gas will be the sum of the specific heats of the constituents.

Substituting the value of mC_v from formula (4), and the value of m, and changing to deg. fahr. we have from the above formulas:

For CO_2, $C_p = 0.19 + .0000977t$ .... a
H_2O, $C_p = 0.426 - .000176t$ .... b
H_2, $C_p = 3.355 - .000678t$ c
CO, $C_p = 0.24 - .0000481t$ .... d
N_2, $C_p = 0.24 + .0000484t$ .... e
CH_4, $C_p = 0.6$ .... f
O_2, $C_p = 0.21 + .0000424t$ .... g

Let a, b, c, d, e and f, represent the mean C_p for the above gases between 32 deg. and t deg. Fahr. Then the C_p of the producer gas = the sum of the products of the constituents of the gas by weight × the specific heat of the constituent.

That is,

$$\text{Item }122 = [a \times \text{Item }144 - c \times \text{Item }147 + d \times \text{Item }145 - e \times \text{Item }152 - f \times \text{Item }148 + g \times \text{Item }146]\,\frac{1}{100}$$

Item 123

1	2	3	4
$CO_2 = O_2 - C$	$CO = C + O$	$CH_4 = C - 2H_2$	$C_2H_4 = 2C + 2H_2$
$44 = 32 - 12$	$28 = 12 + 16$	$16 = 12 - 4$	$28 = 24 - 4$

Total weight of carbon appearing in a unit weight of gas from the above = per cent by weight $CO_2 \times \frac{3}{1100}$ + per cent by weight $CO \times \frac{3}{700}$ − per cent by weight $CH_4 \times \frac{3}{400}$ − per cent by

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[1] Mallard and Le Chatelier's Formulas.

weight $C_2H_4 \times \frac{6}{700}$

The total weight of H_2 appearing in a unit weight of gas = per cent by weight $\frac{H_2}{100}$ + per cent by weight $CH_4 \times \frac{1}{400}$ + per cent by weight $C_2H_4 \times \frac{1}{700}$

or Item 123 = [Item 144 × 0.273 + Item 145 × 0.429 + Item 148 × 0.75 + Item 149 × 0.858] ÷ [Item 147 + Item 148 × 0.25 + Item 149 × 0.143]

Item 124. Observed.

Item 125. Let G = total volume of gas as measured by the meters.

p = absolute pressure of this gas in inches Hg. as observed.

T = absolute temperature in deg. fahr.

t = observed temperature.

The volume of gas G as measured by the meter is saturated with water vapor at the temperature t.

Let p_1 = pressure of this vapor in inches as obtained from the steam table.

Then as the pressure p is the total pressure of the mixture, the actual or partial pressure of the dry gas is $P - p_1 = p_2$.

Let p_s, G_s, and T_s, be the condition of standard gas. Then

$$\frac{G_s \times p_s}{T_s} = \frac{G \times p_2}{T} \text{ or } G_s = \frac{G \times p_2 \times T_s}{T \times p_s} = \frac{G \times p_2 \times 522}{T \times 30} = \frac{G \times p_2}{T} \times 17.4$$

Therefore Item 125 equals

$$\frac{\text{Item } 124 \times \text{Item } 26 \times 17.4}{\text{Item } 45}$$

Item 126. Calculation of the volume of the gas from the analysis of the gas and the analysis of the coal. Evidently the total weight of the carbon appearing in the gas should be equal to the total weight of carbon in the coal minus the weight that is lost through the grate and the weight lost in soot and tar. This latter is small for the hard-coal producer and will be neglected.

Let P = Per cent carbon by weight in dry coal.

W = total weight of dry coal.

W_1 = total weight of ash and refuse.

P_1 = Per cent by weight of carbon in the ash and refuse.

W_2 = total weight of carbon that should appear in the gas, or the weight of carbon utilized in the producer.

$$W_2 = \frac{PW - P_1 W_1}{100}$$

This carbon is contained in the CO_2, CO, CH_4, C_2H_4.

The proportion by weight of C in CO_2 is 3/11, of C in CO is 3/7, of C in CH_4 is 3/4 and of C in C_2H_4 is 6/7.

Therefore the total weight of C contained in a unit weight of gas will be

$$W_3 = \frac{3/11\,A + 3/4\,E + 3/7\,F + 6/7\,G}{100}$$

Where A, E, F, and G are the per cent by weight of CO_2, CH_4, CO, and C_2H_4 from the gas analysis.

The per cent of this carbon contained in the gas as CO_2 is $\frac{3/11\,A}{W_3}$

The actual weight of this carbon will be $\frac{3/11\,A}{W_3 \times 100} W_2$. Since W_2 is the total weight of carbon utilized, from the fuel.

One pound of carbon on burning produces $3\frac{2}{3}$ lb. of CO_2.

$W_2 \times \frac{3/11\,A}{W_3 \times 100} \times 3\frac{2}{3}$ = total weight of CO_2 in the gas.

Let W_s = the specific weight of CO_2 at 62 deg. and 30 in. Hg. See Item 121. The standard volume V_s of CO_2 will therefore be,

$$\frac{A\,W_2}{100 \times W_3 \times W_s} = V_s$$

Let this volume equal a per cent (from the volumetric gas analysis) of the total volume of gas delivered by the producer. The total volume of standard gas from the gas analysis is therefore

$$\frac{100\ V_s}{a} = V^1{}_s$$

$$V^1{}_s = \frac{A \times W_2}{a \times W_3 \times W_s}$$

Item 126 therefore equals

$$\frac{\text{Item 144} \times (\text{Item 51} \times \text{Item 61} - \text{Item 52} \times \text{Item 68})}{0.116 \times \text{Item 135} \times (0.273\ \text{Item 144} + 0.75\ \text{Item 148} + 0.429\ \text{Item 145} + 0.858\ \text{Item 149})}$$

Item 127. Item 126 should be used as a check on Item 125. The difference between the two values should not exceed 5 per cent. Item 125 should be used in all computations.

$$\textit{Item } 128 = \frac{\text{Item 127}}{\text{hours}}$$

$$\textit{Item } 129 = \frac{\text{Item 127}}{\text{Item 51}}$$

$$\textit{Item } 130 = \frac{\text{Item 127}}{\text{Item 54}}$$

$$\textit{Item } 131 = \text{Item 127} \times \text{Item 121}$$

$$\textit{Item } 132 = \frac{\text{Item 131}}{\text{hours}}$$

$$\textit{Item } 133 = \frac{\text{Item 131}}{\text{Item 51}}$$

$$\textit{Item } 134 = \frac{\text{Item 131}}{\text{Item 54}}$$

Item 135 to 143 From chemist.

Items 104. The relative humidity, or per cent saturation is observed by means of a hair hygrometer. This may also be obtained from a wet and dry bulb thermometer, and a set of psychrometric tables.

Item 105. See Kent, page 484 for weights of air and moisture.

Let p = per cent saturation, or relative humidity, Item 104.

n = weight of moisture contained in one cu. ft. of saturated air at the observed temperature, Item 29.

$\frac{pn}{100}$ = weight of moisture in 1 cu. ft. of air as used.

If m = weight of 1 cu. ft. dry air at the observed temperature, then

$$\text{Item 105} = \frac{pn}{100m} \times 100 = \frac{pn}{m} = \text{Item 104} \times \frac{n}{m}$$

This formula is in error due to neglecting the vapor pressure of water; this is, however, negligible in the present case.

Item 82. Observed.

Item 83. Observed.

Item 84. = Item 82 — Item 83.

Item 86. The weight of water decomposed in the producer is evidently 9 times the weight of hydrogen formed, since 1 lb. of water on decomposition yields 1 lb. of hydrogen and 8 lb. oxygen. The total weight of hydrogen formed is equal to the total weight of free hydrogen appearing in the gas, plus the total weight of hydrogen appearing in the CH_4 in the gas, minus the total weight of hydrogen that is not in combination with oxygen in the coal.

Item 86, therefore, equals

$$9 \left[\frac{\text{Item 131 (Item 147} + 0.25\ \text{Item 148)} - \text{Item 51 (Item 62} - \tfrac{1}{8}\ \text{Item 63)}}{100}\right]$$

Item 87. Owing to the difficulty in obtaining the weight of moisture in the gases leaving the producer with a proper degree of accuracy by the use of a dryer, it will ordinarily be better to use Item 86 for this item.

Item 106. Obtained from gas analysis by weight. Items 144 to 152 inclusive.

Let A = per cent CO_2
B = per cent O_2
C = N_2 per cent

Let D = per cent H_2
E = per cent CH_4
F = per cent CO

1	2	3
$C + O_2 = CO_2$	$C + O = CO$	$H_2 + O = H_2O$
$12 + 32 = 44$	$12 + 16 = 28$	$2 + 16 = 18$
$\frac{3}{11} + \frac{8}{11} = 1$	$\frac{3}{7} + \frac{4}{7} = 1$	$\frac{1}{9} + \frac{8}{9} = 1$

From equation (1), one lb. of CO_2 requires $\frac{8}{11}$ lb. of O for its formation

From (2) one lb. CO requires $\frac{4}{7}$ lb, of O for its formation.

The total amount of O appearing in 1 lb of the gas is therefore

$$\left(\frac{8}{11}A + \frac{4}{7}F + B\right) \times \frac{1}{100}$$

This O comes from that contained in the air, that contained in the coal, and from the water decomposed. The oxygen contained in the coal, however, is supposed to be united with hydrogen, and is therefore contained in moisture, which has been allowed for in the water decomposed.

Let W = total weight of gas.

Then the total weight of O used is

$$\frac{W}{100}\left(\frac{8}{11}A + \frac{4}{7}F + B\right)$$

Let W_2 = weight of water decomposed. From (3), 1 lb. of water decomposed liberates $\frac{8}{9}$ lb. of O.

Weight of O supplied by decomposition of water = $\frac{8}{9}W_2$

Let W_3 = total weight of O supplied by air.

From the above equation we have,

$$\left(\frac{8}{11}A + \frac{4}{7}F + B\right)\frac{W}{100} = \frac{8}{9}W_2 + W_3.$$

$$\text{or } W_3 = \frac{W}{100}\left(\frac{8}{11}A + \frac{4}{7}F + B\right) - \frac{8}{9}W_2 \dots\dots\dots\dots (4)$$

The weight of air used is therefore $\frac{W_3}{0.23}$ since the proportion by weight of O in air is 23, or

$$\frac{W_3}{0.23} = W_4 = \frac{1}{0.23}\left[\frac{W}{100}\left(\frac{8}{11}A + \frac{4}{7}F + B\right) - \frac{8}{9}W_2\right] \dots\dots\dots\dots (5)$$

$$\text{Therefore Item 106} = \left[\frac{\text{Item 131}}{100}\left(\frac{8}{11}\text{ Item 144} + \frac{4}{7}\text{ Item 145} + \text{Item 146}\right) - \frac{8}{9} \times \text{Item 87}\right]\frac{1}{0.23} \dots\dots\dots\dots (6)$$

The above computation may be made from the weight of nitrogen appearing in the gas. The nitrogen comes from the air used and from the nitrogen introduced with the fuel.

Let C per cent = weight of N_2 from analysis.

Let W as before = total weight of gas.

Then $\frac{CW}{100}$ = total weight of N_2 in the gas.

The weight of N_2 supplied by fuel will be $\frac{W_1H_1}{100}$, where W_1 equals the total weight of dry coal and H_1 is the per cent by weight of N_2 contained in the coal. We have therefore.

$$\frac{CW}{100} = \frac{W_1H_1}{100} + W_4$$

where W_4 = total weight of N_2 in the air.
The weight of air supplied is therefore

$$\frac{W_4}{0.77} = \left(\frac{CW}{100} - \frac{W_1H_1}{100}\right)\frac{1}{0.77} = W_5 = \left(\frac{CW - W_1H_1}{77}\right)$$

or Item 106 = (Item 131 × Item 152 − Item 51 × Item 64) $\frac{1}{77}$ ....(7)

The weight of air derived by formula (6) will be liable to error, due principally to the error in the determination of the total quantity of water decomposed, which may be large, and also to the neglecting of the SO_2 formed.

The weight determined by formula (7) will be in error due principally to the taking of the weight of N_2 from the analysis by difference.

The results obtained from formulae (6) and (7) should check within 5 per cent.

The results obtained by (7) are believed to be more accurate and will be used in all computations.

Item 107. This may be obtained direct from the calibration curve of the orifice. It should be compared with the two values obtained above.

Item 108. This will ordinarily be taken from Item 106.

Item 109 $= \frac{\text{Item 108}}{\text{Hours}}$

Item 110 $= \frac{\text{Item 108}}{\text{Item 51}}$

Item 111 $= \frac{\text{Item 108}}{\text{Item 54}}$

Item 112 $= \frac{\text{Item 108}}{\text{Item 131}}$

Item 85 = Item 84 + Item 85*b* − Item 85*c*.

Item 85*b* $= \frac{\text{Item 108} \times \text{Item 105}}{100}$

Item 85*c* $= \frac{\text{Item 49} \times \text{Item 50}}{100}$

Item 88 = Item 85 − Item 87

Item 89 $= \frac{\text{Item 87}}{\text{Item 85}}$

Item 90 $= \frac{\text{Item 87}}{\text{Item 131}}$

Item 91 $= \frac{\text{Item 87}}{\text{Item 51}}$

Item 92 $= \frac{\text{Item 87}}{\text{Item 54}}$

Item 93 $= \frac{\text{Item 87}}{\text{Item 108}}$

Item 94 $= \frac{\text{Item 85}}{\text{Item 51}}$

Item 95 $= \frac{\text{Item 85}}{\text{Item 54}}$

Item 96 $= \frac{\text{Item 85}}{\text{Item 108}}$

Item 97 = Observed

Item 98 = Observed

$\textit{Item}\ 99 = \dfrac{\text{Item 84}}{\text{Hours}}$

$\textit{Item}\ 100 = \dfrac{\text{Item 99}}{\text{Item 11}}$

$\textit{Item}\ 101 = \dfrac{\text{Item 87}}{\text{Hours}}$

$\textit{Item}\ 102 = \dfrac{\text{Item 85}}{\text{Hours}}$

$\textit{Item}\ 103 = \dfrac{\text{Item 97}}{\text{Hours}}$

$\textit{Item}\ 115 = \text{Item 114} \cdot \text{Item 121}$

$$\textit{Item}\ 116 = \frac{\text{Item 98} \times 0.2205}{\text{Item 115}}$$

$$\textit{Item}\ 117 = \frac{100\ \text{Item 88}}{\text{Item 131}}$$

Item 153 The grate efficiency is 100 times the ratio of the total B.t.u. in the fuel minus the B.t.u. in the fuel lost through the grate; to the total B.t.u. contained in the fuel. Therefore

$$\textit{Item}\ 153 = \frac{\text{Item 51} \times \text{Item 78} \times 100 - \text{Item 52} \times \text{Item 68} \times 14540}{\text{Item 51} \times \text{Item 78}}$$

Item 154. The hot gas efficiency is 100 times the ratio of the total heat of combustion of the gas, plus the sensible heat of the dry gas, plus the total heat contained in moisture in the gas to the heat of combustion of the dry coal, plus the heat given by the entering air, by the coal as sensible heat, and by the moisture or steam supplied in air.

$$\textit{Item}\ 154 = 100 \left\{ \text{Item 119} \times \text{Item 127} + \text{Item 122} \times \text{Item 131}\ (\text{Item 39} - 62^{\circ}.) + \text{Item 88}\ [1116 + 0.6\ (\text{Item 39} - 212)] \right\} \div \text{Item 51} \times \text{Item 78} + \text{Heat supplied in steam.}$$

The heat given the producer by air, and sensible heat in coal may be neglected if the room temperature is within 20° of the standard temperature 62°. With a producer of the contained vaporizer type or one which utilizes the sensible heat of the gas to make the steam the term "Heat supplied in steam" drops out of the equation.

Item 155. The cold gas efficiency is 100 times the ratio between the total heat of combustion of the gas, to the total heat of combustion of the coal plus heat supplied from outside sources. That is, $\text{Item 155} = 100 \dfrac{\text{Item 119} \times \text{Item 127}}{\text{Item 51} \times \text{Item 78}}$

The efficiency based on 100 % grate efficiency $= \dfrac{\text{Item 155}}{\text{Item 158}}$

$$\textit{Item}\ 157 = \frac{\text{Item 156} \times \text{Item 49}}{0.02 \times \text{Item 127}}$$

$$\textit{Item}\ 158 = \frac{\text{Item 97} \times 1000}{62.5 \times \text{Item 127}} = \frac{\text{Item 97}}{0.0625 \times \text{Item 127}}$$

HEAT BALANCE

DEBIT

Item a. Obtained from Item 78.

Items b, c, d, e, f. Using as a standard the temperature of 62° F., the heat given to the producer by the items *b* to *f* inclusive is in most cases negligible. The error at a temperature of 100° F. is less than 1 per cent for a producer of the contained vaporizer type. However, the formulas will be given for computation of these items.

$\textit{Item b} = \text{Item 110} \times 0.24\ (\text{Item 30} - 62^{\circ}\ \text{F.})$

$\textit{Item c} = \dfrac{\text{Item 85}b \times (\text{H} - 1070)}{\text{Item 51}}$ where H = the total heat in 1 lb. saturated steam at the temperature of the fire room.

$$\textit{Item } d = \frac{\text{Item 49} \times \text{Item 50}}{100 \times \text{Item 51}}\,(\text{Item 30} - 62^\circ\ \text{F.})$$

$$\textit{Item } e = 0.24 \times (\text{Item 30} - 62^\circ\ \text{F.})$$

$$\textit{Item } f = \frac{\text{Item 82}\,(\text{Item 33} - 62^\circ\ \text{F.})}{\text{Item 51}}$$

CREDIT.

$$\textit{Item } a = \text{Item 122} \times \text{Item 133} \times (\text{Item 39} - 62^\circ\ \text{F.})$$

$$\textit{Item } b = \frac{\text{Item 117} \times \text{Item 133}}{100}\,[(\text{Item 39} - 212^\circ\ \text{F.}) \times 0.6 + 1116]$$

$$\textit{Item } c = \text{Item 119} \times \text{Item 129}$$

$$\textit{Item } d = \frac{\text{Item 52} \times \text{Item 68} \times 145.40}{\text{Item 51}}$$

Item e This is very small and may be neglected.

$$\textit{Item } f = \frac{\text{Item 83}}{\text{Item 51}}(\text{Item 34} - 62^\circ\ \text{F.})$$

Item g = Sum of Items on debit side − (Item *a* + Item *b* + Item *c* + Item *d* + Items *e* and *f*.)

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*_Circular No. 1._ High-Speed Tool Steels, by L. P. Breckenridge. 1905.

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*_Circular No. 2._ Drainage of Earth Roads, by Ira O. Baker. 1906.

*_Circular No. 3._ Fuel Tests with Illinois Coal. (Compiled from tests made by the Technologic Branch of the U. S. G. S., at the St. Louis, Mo., Fuel Testing Plant, 1904-1907.) by L. P. Breckenridge and Paul Diserens. 1909

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*_Bulletin No. 7._ Fuel Tests with Illinois Coals, by L. P. Breckenridge, S. W. Parr and Henry B. Dirks. 1906.

*_Bulletin No. 8._ Tests of Concrete: I. Shear; II. Bond, by Arthur N. Talbot. 1906.

*_Bulletin No. 9._ An Extension of the Dewey Decimal System of Classification Applied to the Engineering Industries, by L. P. Breckenridge and G. A. Goodenough. 1906.

*_Bulletin No. 10._ Tests of Concrete and Reinforced Concrete Columns, Series of 1906, by Arthur N. Talbot. 1907.

*_Bulletin No. 11._ The Effect of Scale on the Transmission of Heat through Locomotive Boiler Tubes, by Edward C. Schmidt and John M. Snodgrass. 1907.

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*_Bulletin No. 17._ The Weathering of Coal, by S. W. Parr, N. D. Hamilton, and W. F. Wheeler. 1908.

*_Bulletin No. 18._ The Strength of Chain Links, by G. A. Goodenough and L. E. Moore. 1908.

*_Bulletin No. 19._ Comparative Tests of Carbon, Metallized Carbon and Tantalum Filament Lamps, by T. H. Amrine. 1908.

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Bulletin No. 39. Tests of Washed Grades of Illinois Coal, by C. S. McGovney. 1909.

Bulletin No. 40. A Study in Heat Transmission, by J. K. Clement and C. M. Garland 1910.

Bulletin No. 41. Tests of Timber Beams, by Arthur N. Talbot. 1910.

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BULLETIN NO. 51 DECEMBER 1911

STREET LIGHTING

BY J. M. BYRANT, ASSISTANT PROFESSOR OF ELECTRICAL ENGINEERING AND H. G. HAKE, INSTRUCTOR IN ELECTRICAL ENGINEERING

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STREET LIGHTING

I. INTRODUCTION

1. *Preliminary.*—It is the purpose of this bulletin to make available information concerning street illumination. The suggestion which led to this compilation came from the many inquiries received by the Electrical Engineering Department each year from those interested in framing ordinances permitting corporations or individuals to operate street lighting systems. An attempt has been made to present this information in such a form as to be readily understood by the general public, without requiring any special technical knowledge. The data have been compiled from reliable sources, and checked in many instances by tests conducted by the writers.

This bulletin is designed to be of assistance to central station superintendents, and to the general public in selecting the proper lamp and fixing the charge for the same. It is also designed to be of value to the illuminating engineer and to the manufacturer, and in clearing up, or perhaps in preventing misunderstandings, which so frequently arise between municipalities and power companies. These misunderstandings, being usually the outcome of imperfect knowledge of the effectiveness and limitations of ordinary street lighting units, may readily be prevented if specifications and contracts be made definite and clear.

Many contracts are based upon the rates charged in cities of about the same size for the same class of service, regardless of the comparative operating cost of the installation in each case. Many companies do not compute the cost of any one particular branch of their service, but are satisfied if their total yearly balance shows a profit. Other companies, while knowing the cost of operation of the street lights, are still governed by the prevailing rate in similar cities. Thus it is that some companies may furnish the light to the city streets at an actual loss, but are compensated for this loss by special privileges in other branches of the service. It is believed by the writers that fewer misunderstandings would arise between the consumer and the company if each branch of service received its proper compensation.

This bulletin is also designed to show that the same type of light will not serve equally well for the illumination of all classes of city streets; and that certain accepted standard spacings of lights can never produce adequate and economical street illumination.

II. Production of Light

2. *General Theory.*—Probably the most familiar form of illuminant is the flame. At least, this is the earliest historical form. In the flame, the light is produced by small particles of carbon being brought to a temperature at which they will unite with the oxygen of the air. The heat which gives to these particles their incandescence comes from the union of the various compounds in the combustible with the oxygen. If the flame is cooled in any manner, the carbon will deposit in the form of soot before it reaches a sufficient temperature to burn. If enough air be mixed with the gas formed by the heated illuminant to consume the compounds, an almost non-luminous flame results. Thus the alcohol flame is nearly non-luminous; as is also that of the oxyhydrogen flame and the Bunsen burner. It is well-known that non-luminous flames may be made luminous by the introduction of some foreign substance which need not necessarily be consumed; of such character is the Welsbach burner.

In the Welsbach burner, another interesting phenomenon takes place. If the substance being heated is of a certain character, which is represented by carbon, a combination of colors is obtained for a given temperature. This is known as white light. If, however, the substance be some one of the metals, certain of the colors will be more pronounced; i. e., not all substances give off the same color when heated to the same temperature in a flame. This phenomenon is known as selective radiation. Use is made of this in the newer forms of illuminants in producing more light for a given amount of power or of illuminant. In the Welsbach burner, thoria and seria are spread upon a thin web of heat resisting material. These compounds, when heated to the high temperature of the gas flame, give off a much greater amount of light than would be given by carbon at the same temperature. Many other metals display this property to a greater or less extent. In this field, lie the recent advances in the development of arc lights.

3. *The Incandescent Lamp.*—In the incandescent lamp, light is produced by a fine filament or wire of some material, heated to

incandescence by a current of electricity. Formerly, all such lamps were made with filaments of carbon. These were followed by metallized filaments, and later by tantalum and tungsten, the latter being now used in the form of a drawn wire. All of these lights have their filaments in a vacuum to prevent rapid oxidation of the material, and also to prevent the heat from being conducted away from the filament as rapidly as it would be in air or any dense gas. Such lamps are made for house lighting and street lighting.*

4. *The Arc Lamp.*—(*a*) If two pieces of conducting material be connected in an electrical circuit with their ends touching, and these ends be separated slowly, a spark will be seen between them. If the voltage of the circuit be sufficient, portions of the terminals will be burned away, forming a vapor between them. This vapor is in itself a conductor, and current will continue to flow, even though the terminals are rather widely separated. Light will also be given off from the electrodes and the arc stream, thus giving the arc light.

In the earlier forms of arc lamp, plain carbon rods were commonly used for the terminals, or electrodes, as they are called. The voltage to be used across a given lamp, and the current necessary, are so determined for each type as to produce the proper characteristics. The old direct current open arc lamp required about 50 volts and took 9.6 amperes. This lamp was known as the "full arc" or 2000 "nominal candle-power" lamp. "A half arc" lamp using 6.6 amperes, known as the 1200 "nominal candle-power" lamp, was also used to a considerable extent. In any given type of lamp, the amount of light and the consumption of the carbons depend upon the current used and the size of the carbons. The light given off by an arc light proceeds partly from the electrodes and partly from the vapor or arc stream.

(*b*) In the direct current open arc, most of the light comes from a crater formed in the positive electrode. The upper electrode is made positive to throw most of the light in a downward direction, i. e., in the lower hemisphere. Only about five per cent of the light comes from the arc stream. The voltage across such a lamp must be increased when the arc is lengthened, but not directly in proportion to the distance between the electrodes. This is due to the fact that nearly a constant value of electromotive-force is consumed in vaporizing the electrode material.

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*Tests on multiple burning lamps of different classes have been made by the Electrical Engineering Department, and reported in Engineering Experiment Station Bulletins No. 19 and 33.

Since any energy used in the arc stream produces but little light and lowers the efficiency, it is desirable to use as short an arc as possible without cutting off too much of the light. A long arc is also more difficult to maintain steady on account of the influence of air currents The electrodes in such an arc are consumed rapidly; the positive being consumed about twice as fast as the negative. From 1.5 to 2.0 in. of positive electrode are consumed each hour. Such a lamp must be trimmed each day, i. e., the electrodes must be replaced, entailing expense in labor and material.

(*c*) The rapidity of the consumption of the electrodes in an open arc is due to their oxidation in the air. In order to lengthen the life, or number of hours of burning, of the electrodes, and decrease the expense due to trimming, the enclosed arc lamp was invented. In such a lamp, the arc is produced as in the open arc between carbon electrodes. It is, however, enclosed in a nearly air-tight chamber of glassware known as the inner globe, to distinguish it from the much larger outer globe used on nearly all arc lamps. The air being excluded the electrodes are consumed slowly, their life averaging about 100 hrs., or about 10 times that of the open arc. However, more expensive electrodes must be used, since they must fit quite closely into a gas cap at the top of the inner globe, and yet be fed by the mechanism with perfect freedom. Forced carbons are used in enclosed arcs and moulded carbons in open arcs. Enough air must be introduced into the inner globe of such a lamp to consume all the carbon volatilized by the arc, in order that it shall not deposit on the interior of the inner globe and shut in the light, lowering the efficiency of the lamp.

(*d*) Later improvements in the production of light from an arc lamp consist in securing more light from the arc stream. This has been accomplished by impregnating the electrodes with some metallic salt. The light from an arc stream of pure carbon is purplish in color, but since it comprises such a small proportion of the total light from the lamp, its effect is scarcely noticeable except when colors are being matched. Salts of certain metals, such as calcium, strontium, barium, titanium, sodium, etc., give their own characteristic color to the light, raising its efficiency by luminescence. These salts tend to make the arc unsteady or flickering, necessitating the introduction of other compounds to steady the light. Arc lamps using impregnated electrodes are known as flame arcs. When magnetite is used for one of the electrodes, the lamp is called a magnetite or luminous arc, the latter being a trade name. Magnetite, containing iron, furnishes the conducting material in the arc stream. Most of the light is produced by a compound containing titanium, which is mixed with the magnetite. The electrode consists of a thin iron tube into which ground magnetite, titanium oxide, and chromite are packed. The magnetite melts, volatilizes, and carries with it the titanium. The chromite is used to absorb the fluid magnetite and steady the arc.

The electrodes in a flame arc lamp are consumed very rapidly. Even with long electrodes (14 to 18 in.), they do not last more than one night, or about 10 to 17 hours. The magnetite electrode, on the other hand, lasts from 85 to 200 hours, depending upon the current used. Since the electrodes contain considerable quantities of foreign impregnating materials which are not consumed in the arc, it is impossible to apply an ordinary inner globe to such a lamp. In fact, a draft tube or chimney is usually employed to carry away the products so that they are not deposited on the outer globe. It is obvious that such an opaque deposit would shut in most of the light after the lamp had operated a short time.

In the long burning flame arc, an attempt has been made to partially enclose the arc. Such a lamp has a fairly close fitting inner globe and a condensing chamber. The gases pass first from the arc to the cool condensing chamber, where they deposit the foreign material. They return along the sides of the inner globe by natural draft. Otherwise the operation is similar to that of the enclosed arc lamps. The electrodes have a life of about 70 hours with ordinary current densities.

III. Systems of Distribution

5. *Series.*—There are two systems of electrical distribution, series and multiple. The series system is almost universal for street lighting. Either direct or alternating current may be used on either of these systems. Certain types of arc lamps will operate on either direct or alternating current, others on direct current only. In the direct current system, the current always flows in the same direction, i. e., from the positive terminal of the generator through the external circuit, and back to the negative terminal of the generator. In the alternating current system, the direction of flow of current changes very rapidly, and the electrodes of a lamp must change in polarity at the same time as

that of the circuit. In a 60-cycle alternating current circuit, the polarity changes 120 times per second. It is obvious that the current must be zero at some instant during the change. Hence the conducting path must have a low enough resistance to allow the current to pass again easily. In order to produce sufficient conducting material for an alternating current arc, one or both of the electrodes is made hollow, and filled with a softer and more volatile substance in the form of a continuous core. These electrodes are slightly more expansive than those used in the direct current lamps.

For street lighting, lamps are usually operated in series, i. e., the same current passes from the station through all the lamps in one group or circuit and back to the station. Apparatus is provided at the station to keep the current constant for each circuit of lamps. The mechanism of the lamp regulates the arc length and holds the voltage across the arc constant. This mechanism must also be able to start the lamp at any time, or to cut it out of service without breaking the circuit, if, for any reason, the lamp is inoperative, as any break in the circuit will cut out all the lamps of that circuit.

6. *Multiple.*—The first machines for supplying electrical energy from central stations were for the production of direct current. These machines were nearly always made to supply power to incandescent lamps at a low potential, about 100 volts. They were unsuitable for operating more than two arc lamps in series. Accordingly, direct current arc light generators were devised. Many of the early types are still in use with very little change from the first design. When only direct current is available from a station, it is necessary to provide a separate machine for the arc lights, since there is no means of changing low voltage direct current to a higher voltage. These machines can not be used to assist the low voltage machines in times of emergency. They thus form an additional investment and source of expense to the station for repairs.

Upon the introduction of alternating current generators and systems of distribution, direct current was driven out of the field where power must be transmitted for any considerable distance, on account of the fact that alternating currents may be transformed from one voltage to another readily and with very little loss. With low voltage distribution, a large investment, and consequent high fixed charge is required, due to the heavy copper conductors necessary for efficient transmission. Since the greater part of the load on most central stations is due to incandescent lamps operated on multiple circuits, it was found desirable to install apparatus of constant voltage for their supply. Alternating current arc lamps were readily devised, and the direct current arc light generators were replaced by regulators or by constant current transformers, thus reducing the cost of station equipment.

Alternating current arc lamps are not as efficient light producers as direct current arc lamps, but the combination of alternating current arc lamps and regulators is more desirable than separate direct current arc light machines. Recently, rectifiers have been invented to convert constant alternating current into constant direct current, so that nearly any type of lamp may be operated with fair efficiency and continuity of service from an alternating current constant potential generator. This has led to increased activity in the perfection of direct current arc lamps for street lighting. Thus, to operate constant alternating current arc lamps from a central station supplying constant potential alternating current for multiple incandescent lighting, there will be required additional equipment of arc light transformers or regulators. To operate direct current arc lamps from such a station requires the same equipment as for alternating current lamps and the addition of a rectifier outfit for each group of lamps; or separate direct current arc light generators must be installed. Series incandescent lamps may be operated on either of these circuits.

A standard voltage of 110 volts has been adopted for use on multiple lighting circuits. Since arc lamps by their characteristics require considerably less voltage, some device must be used in series with the arc on multiple arc lamps to limit the current to the proper value. Any such device entails a considerable loss. In D. C. lamps, this loss is proportional to the differences between the line and arc voltage, multiplied by the current flowing. When the line voltage is a little greater than twice the arc voltage two arcs may be operated in series. This method is not satisfactory, since if one arc goes out, it extinguishes the other in series with it, or some complicated mechanism must introduce a resistance capable of absorbing the same voltage as one arc lamp.

In an A. C. system, a device known as a reactance coil may be used to reduce the voltage, at the same time absorbing only a nominal amount of power, not over 10% of the lamp watts. Such a device, however, introduces other undesirable factors into the circuit.

The reason for using series instead of multiple lights, is partly due to the less efficient lamps, and partly to the larger and much more expensive conductors required for the multiple system. The current for each group of lamps may be carried over a rather small size of conductor, without regard to the number of lamps in circuit, if they are all connected in series. A No. 8 B & S gauge wire would be of sufficient current carrying capacity. Some companies, however, use No. 6 or No. 4 on account of the greater mechanical strength when open wiring is used. If the same number of lamps were fed over a multiple circuit, several times the amount of money would have to be invested in copper, making a more expensive and less efficient installation.

IV. PHOTOMETRY AND ILLUMINATION

7. *Photometry.*—The accurate photometric measurement of the intensity of light from an arc lamp is very difficult to make even with the facilities of the best laboratories. This is due to the unsteadiness of the light. Devices have been made for determining the mean spherical intensity of such lights at one setting. These devices are, however, very expensive, and are beyond the means of most central station laboratories. The distribution curves, as published by many different writers, are usually quite reliable. They are sufficiently accurate for calculating street illumination. Such curves are illustrated in the following pages.

It must be remembered in these comparisons that most curves are taken in a laboratory where the lamp is kept in proper adjustment throughout the test, and the glassware carefully cleaned. The actual illumination received from such lights, when in continuous service, and with ordinary care in trimming, does not average over 80 per cent of the laboratory values. This loss of light is due to dirt on the glassware, deposit on the inner globe of enclosed lamps, and lack of adjustment of the mechanism due to wear, corrosion, etc. On the whole, illumination readings of actual installations are much more reliable, if made with sufficient care.

8. *The X Value.*—There is great need of a new standardization of the method of specifying street illumination in this country. In 1907, the National Electric Lamp Association adopted a standard specification for street lights. Comparative tests of the illumination thrown upon the street, at distances of between 200 and 300 ft. from the arc, were made upon the various street lights

then in use. From these values, a factor known as the "X" value of the lamp was computed. The X value shows the relation between the illumination from the lamp in question, and that given by a standard 16 candle power incandescent lamp at 1/X of the distance. Thus, for example, an arc lamp, having an X value of 4, gives the same illumination as a 16 candle power incandescent lamp at $\frac{1}{4}$ the distance, and a lamp having an X value of 5 gives the same illumination as the 16 candle power incandescent lamp at $\frac{1}{5}$ the distance. Thus a lamp which has an X value of 5, at a distance of 250 ft., gives the same illumination as a 16 candle power incandescent lamp, at a distance of 50 ft. With the advent of the demand for brighter illumination in this country, necessitating closer spacing of the lamps, this specification is no longer a measure of the value of the lamp. Other methods of specification used in different countries, or by different engineers, include horizontal illumination on a plane about four feet above the street surface, illumination on a vertical plane at a given point, and illumination on a plane normal to the rays of light from the lamp and at various distances.

As will be shown later, the illumination on the street surface depends not only upon the light at a certain angle, but upon the distribution of light about the arc at all angles, and upon the height and spacing of the lamps. The specification of the illumination should also include the distribution of light about the lamp for a given condition. The amount and character of the illumination required depends upon the character of the street surface and the use made of the street. Thus a lamp with a good X value may be suitable for lighting one street, but wholly unsuitable for another.

9. *Measurement of Illumination.*—Since the light from any one arc lamp varies continuously on account of the arc shifting about on the ends of the electrodes, it is difficult to arrive at accurate results in the measurement of street illumination. In thoroughfares where several arcs per pole or incandescent lamps are used, the illumination is much more steady. The illumination may be measured by some form of illuminometer. Where accurate results are not necessary, some form of reading photometer may be used. Such an instrument is shown in section in Fig. 1. The observer stands with his back toward the light, and looks into the photometer through the aperture A, his eyes being shielded from other lights. The illumination to be measured is that upon the surface C, which is a card containing lines of letters

or characters of varying boldness. Such a card is illustrated in Fig. 2. The light which falls on the surface C comes into the photometer through the tube B. The line of type on the card is so selected as to show the small characters dimly or blurred, while the large characters and capitals remain clear-cut. The distance from the observer to the light is also measured.

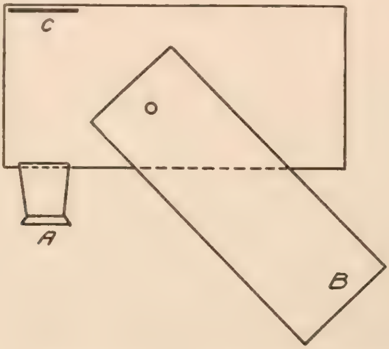

FIG. 1. SECTION OF A READING PHOTOMETER

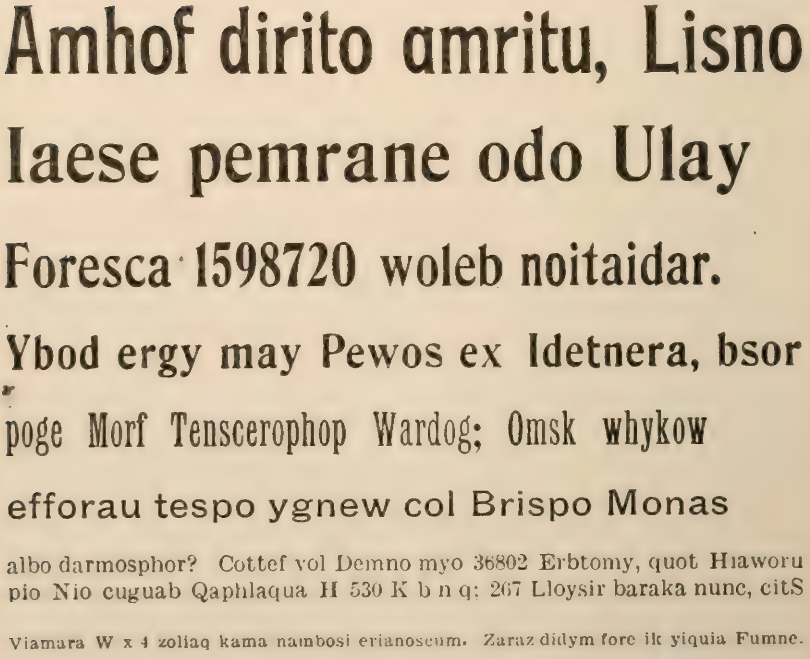

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FIG. 2. SAMPLE CARD FOR READING PHOTOMETER

By previously calibrating the instrument by comparison with a standard lamp, values of illumination may be determined with an accuracy of from 5 to 10 per cent, depending upon the intensity of the illumination to be measured. Calibration is performed by finding the distance from the standard light at which the different lines of characters may be read. The illumination in foot candles may be computed from the candle power of the lamp and from the distance in feet as follows. Suppose the lamp giving 16 candle power to be in a horizontal position, and suppose the distance from the lamp to be 4 ft. Then since the illumination, from a small object as a source, varies inversely as the square of the distance, the illumination would be $\frac{16}{4^2} = 1$, or one foot candle. This instrument measures the illumination on a normal plane. To reduce the illumination to that on a horizontal plane, these values must be multiplied by the cosine of the angle between this plane and the street surface at that point. This form of illuminometer is inexpensive and may be constructed in any carpenter shop. Other more expensive illuminometers are on the market which are more accurate, but are also more difficult to operate. Unless there is considerable reflection from buildings, the street illumination may be calculated quite approximately from the light distribution curve of the lamp, its height and the distance from its base to the point considered, as will be explained later.

10. *Distribution Curves.*—On the following pages will be found the distribution curves of the more common lamps used for street lighting. Data for the distribution curves are obtained by measuring the candle power at different angles in a vertical plane about the lamp by means of a photometer. In such a device, the lamp to be measured is compared with a standard lamp. This comparison is difficult to make for an arc lamp on account of the high intensity of the light in some directions, and the continual shifting about of the arc on the electrodes, with the consequent change in the distribution. The color of the light, in some instances differing from that of the comparison lamp, makes the determination more difficult. Special and expensive apparatus must also be used for obtaining the intensity at different angles. The results of such determinations are shown in Fig. 3, 4, 5 and 6. A complete circle is divided into 360°, and radii are drawn at convenient angles. On these radii, the intensity in candle power is laid off at convenient scale. The points thus determined are connected by a smooth curve known as the "distribution curve". The candle power in any direction in a vertical circle is the length of the

radius, at that angle, intercepted by the curve, multiplied by the candle power scale. The distribution in a horizontal plane through the source is usually a circle; so that the distribution in one vertical plane represents the lamp distribution with sufficient accuracy.

Since, in street lighting, only the light thrown downward, i. e., in the lower hemisphere, is useful for illumination, only that portion of the curve is shown. All light in the upper hemisphere is wasted unless a reflector is used to divert these beams into useful directions. In order to save space and allow a larger scale, only one-half the lower hemisphere is shown in this bulletin, the other being exactly like it.

In Fig. 3, A is the distribution curve from a 9.6 amperes, direct current open arc lamp. It will be seen that the maximum

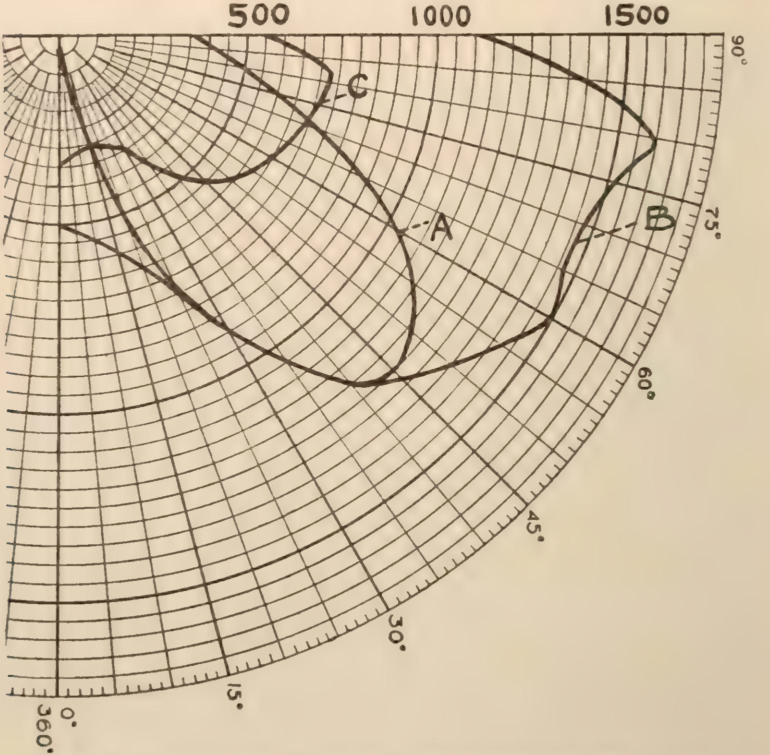

Fig. 3. Distribution Curves of Arc Lamps with Clear Outer Globes

A. 9.6 Amperes, D. C. Open Arc.
B. 6.6 Amperes, D. C. Magnetite Arc.
C. 4 Amperes, D. C. Magnetite Arc.

intensity is at 45° from the vertical axis. The ray in this direction will strike the ground at a distance equal to the height of the lamp above the street surface. The light from this lamp comes mostly from the crater in the upper or positive electrode. The arc being short, the shadow cast by the lower electrode will cover a considerable angle, as seen in Fig. 7. The source of light being a small area, its intrinsic brilliancy, or candle power per sq. in., is of necessity very high. Shadows from the side rods and lower carbon of the lamp are also more or less annoying, because they are so much magnified on the surface of the ground.

In the enclosed direct current arc lamp, the curve for which is shown in Fig. 4 as C, the same shadow from the lower electrode appears, but it is not so intense on account of the fact that the curve is modified by the presence of the inner globe; the arc also

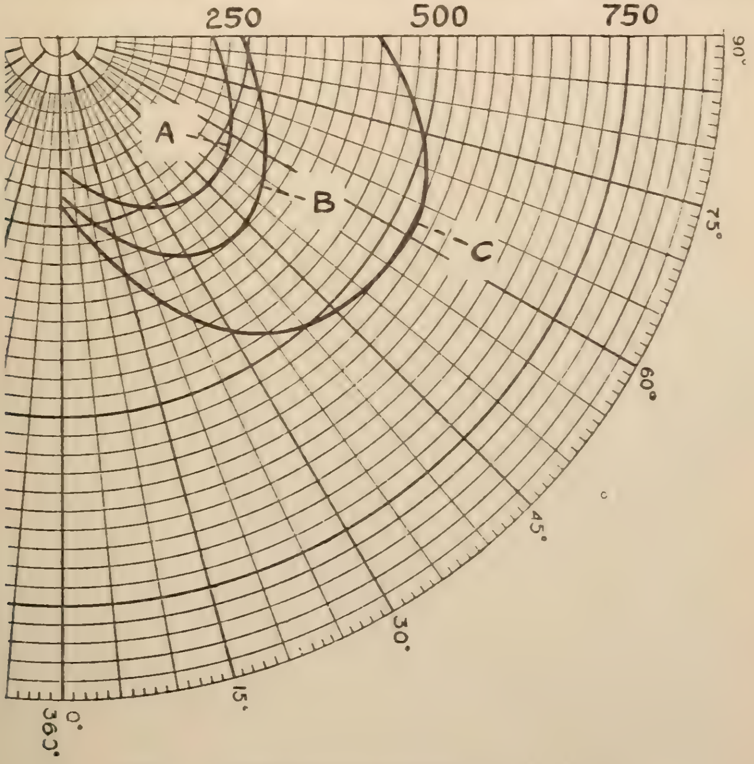

FIG. 4. DISTRIBUTION CURVES OF ARC LAMPS

A. 6.6 Amperes A C. Enclosed, Opal Inner. Clear Outer Globe.
B. 7.5 Amperes A. C. Enclosed, Opal Inner, Clear Outer Globe.
C. 6.6 Amperes D. C. Enclosed. Opal Inner, Clear Outer Globe.

is longer. The globe absorbs some of the light, and also becomes a secondary source of light, making the intensity less in the maximum direction and greater in other directions and at wider angles.

In the alternating current enclosed arc lamp, as much light is thrown in the upper hemisphere as in the lower, since craters are formed alternately in the upper and the lower electrodes. A reflector or shade should be used always with such a lamp, to return as much of the light as possible to the lower hemisphere. Two curves for such lamps are shown as A and B in Fig. 4, A being for a 6.6 amperes lamp, and B for a 7.5 amperes lamp.

In the flame arc lamp, the distribution curve depends upon whether the arc is formed between vertical electrodes, as shown in Fig. 8, or between inclined electrodes as shown in Fig. 9. In the latter case, the light will all be in a downward direction and strongest immediately below the light, as in B, Fig. 5. When

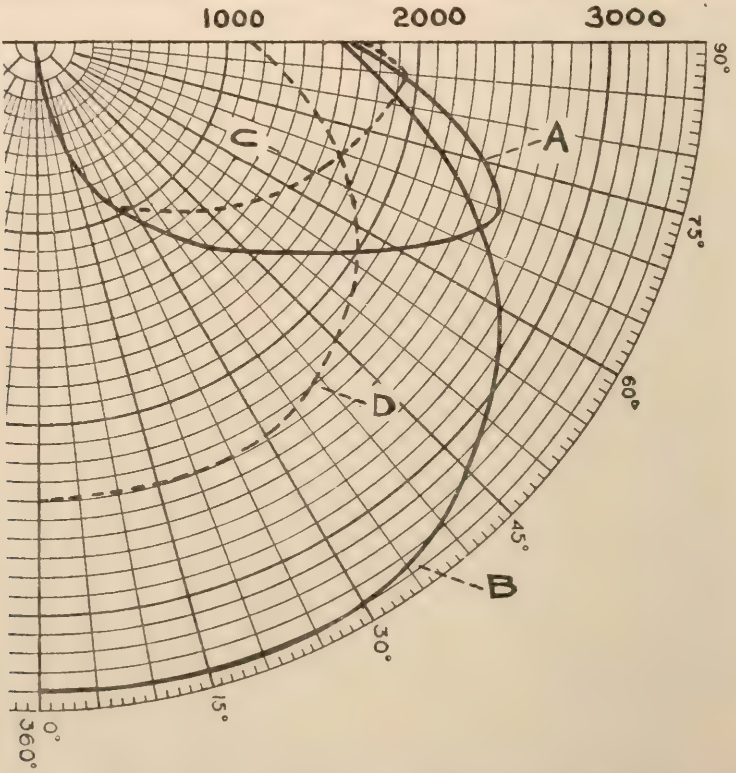

FIG. 5. DISTRIBUTION CURVES OF ARC LAMPS

A. Long Burning Flame, 5.5 Amperes. D. C., Clear Inner and Clear Outer Globe.
B. Inclined Electrode Short Burning Flame, 10 Amperes D. C., Clear Outer Globe.
C. Long Burning Flame, 5.5 Amperes, D. C., Clear Inner, Opal Outer Globe.

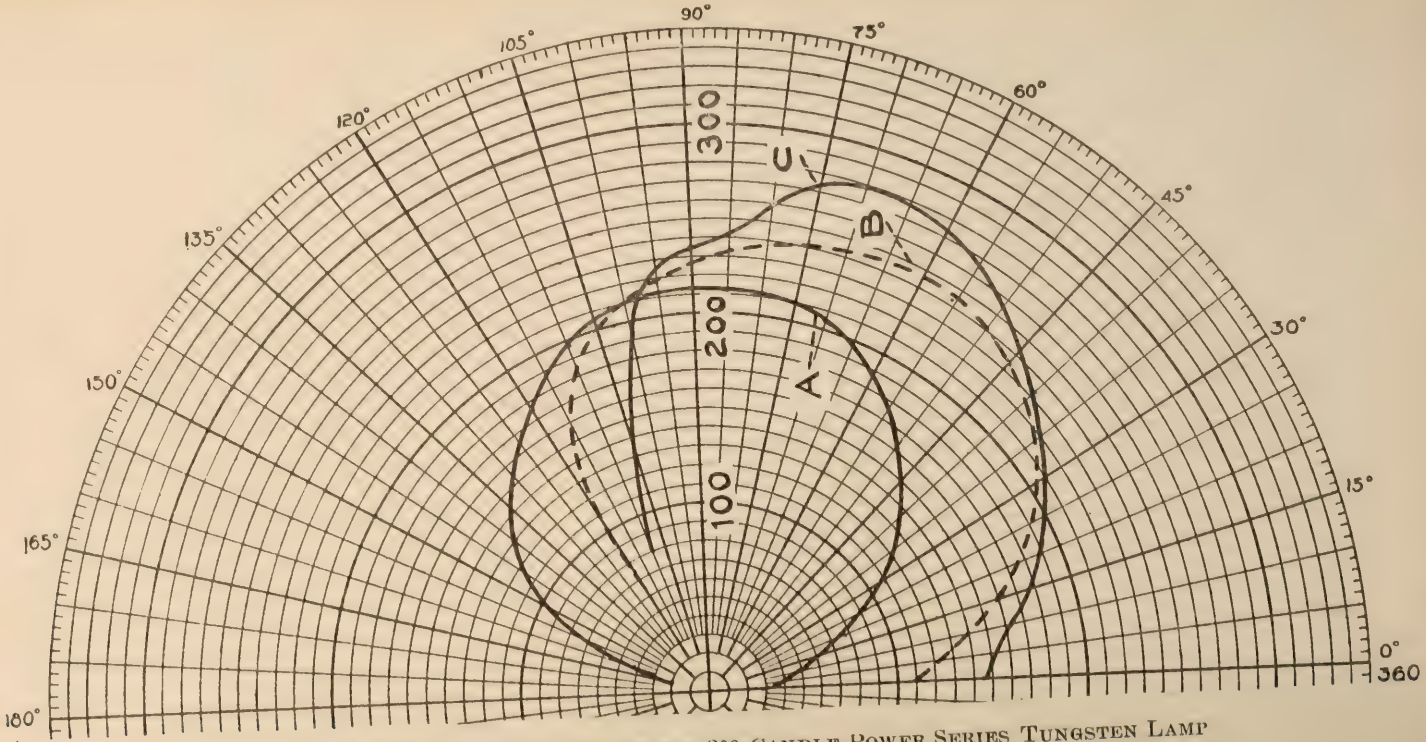

FIG. 6. DISTRIBUTION CURVE FOR 200-CANDLE POWER SERIES TUNGSTEN LAMP

A. Bare Lamp.
B. Lamp with 22 inch Enamelled Reflector.
C. Same with Radial Wave Reflector.

vertical electrodes are used, the light will be about equal in each hemisphere, but may be modified by use of the shade.

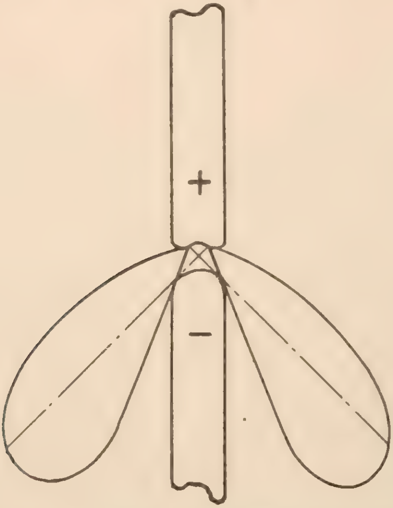

FIG. 7. TYPICAL DISTRIBUTION CURVE OF D. C. OPEN ARC LAMP

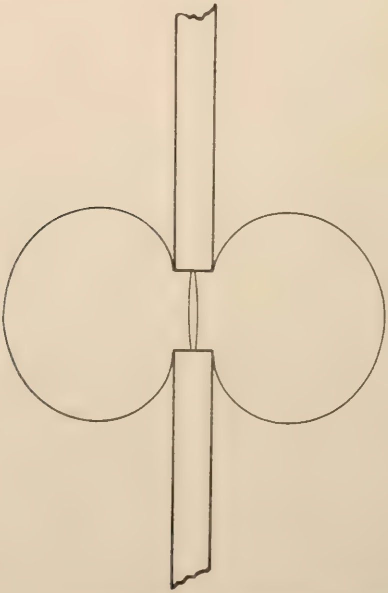

FIG. 8. TYPICAL DISTRIBUTION CURVE OF VERTICAL ELECTRODE, FLAME ARC LAMP

The distribution curve for a tungsten series incandescent light is similar in shape to that shown in Fig. 8, or in A, Fig. 6, which is the distribution curve for a rated 200 candle-power lamp. Various shades and reflectors have been developed for use with these lamps. They vary from a 22-in. enamel reflector, giving a distribution as shown in B, Fig. 6, or the radial reflector with its curve C, Fig. 6, and various unsymmetrical reflectors throwing most of the light into the middle of the street and very little on the sidewalk. For this latter shade, the light must be supported at the side of the street.

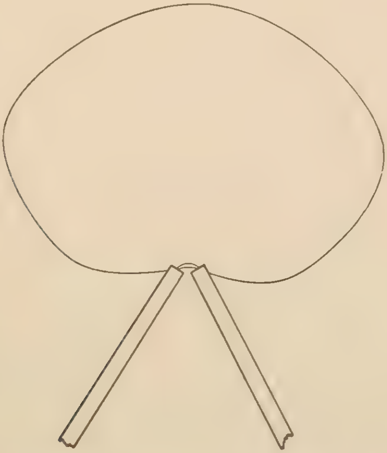

FIG. 9. TYPICAL DISTRIBUTION CURVE, INCLINED ELECTRODE, FLAME ARC LAMP

11. *Effect of Shades and Reflectors.*—That the distribution curves of lamps may be greatly altered by the use of reflectors and globes has already been mentioned To illustrate the effect of a reflector, two curves of the same lamp are shown in Fig. 10. Curve A is a distribution curve for a long burning flame arc lamp with clear inner globe and thin opal outer globe but with no reflector. Curve B is for the same lamp, with the same globes and with a 24-in. white porcelain enameled reflector added. The shape of the distribution curve has thus been altered to suit a certain condition. The mean lower hemispherical candle-power has been increased by about 5 per cent, but the mean spherical candle-power has been decreased by 24 per cent. In arc lamps using stationary lower electrodes, the shape of the distribution

curve, with reflector, changes during the life of the electrodes, due to the change of position of the source of light. Fig. 11 illustrates the effect of globes on the distribution curve of the lamp. Curve A is for a vertical carbon flame arc lamp with 26 in. reflector and with clear outer globe. Curves B and C are for the same lamp, with the same reflector, but with opal globes of different densities. Nearly the same shape of distribution curve has been obtained by both of the globes. However, 10 per cent of the light has been absorbed by one of the globes and 25 per cent by the other. Thus it is seen that an opal globe tends to make the distribution curve more uniform, but in so doing, it

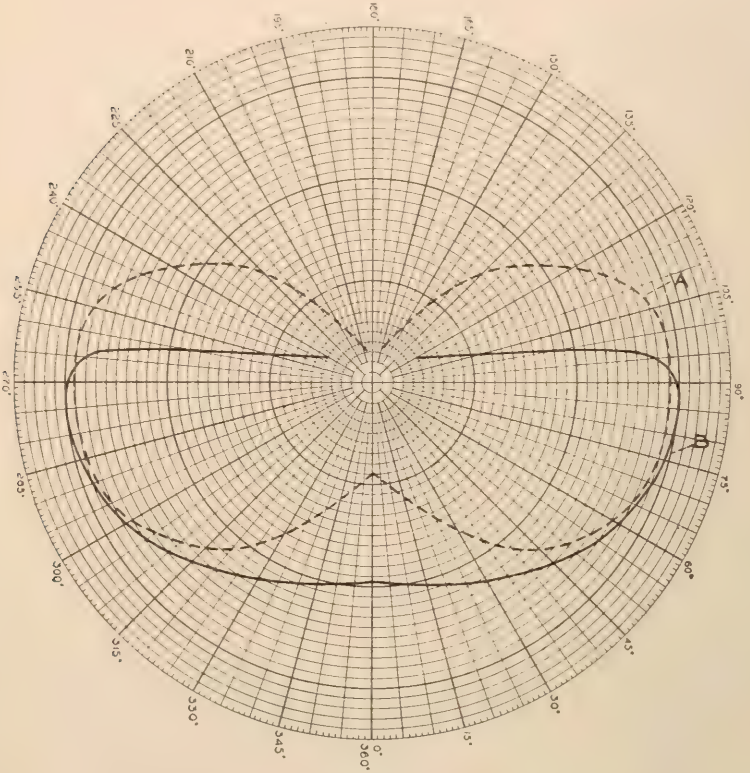

FIG. 10

A. Distribution Curve of the Long Burning Flame Arc Lamp with Opal Globe, No Reflector.
B. Same Lamp with Reflector.

absorbs a considerable portion of the light. This type of globe also lowers the intrinsic brilliancy, since it becomes, in effect, the source of light.

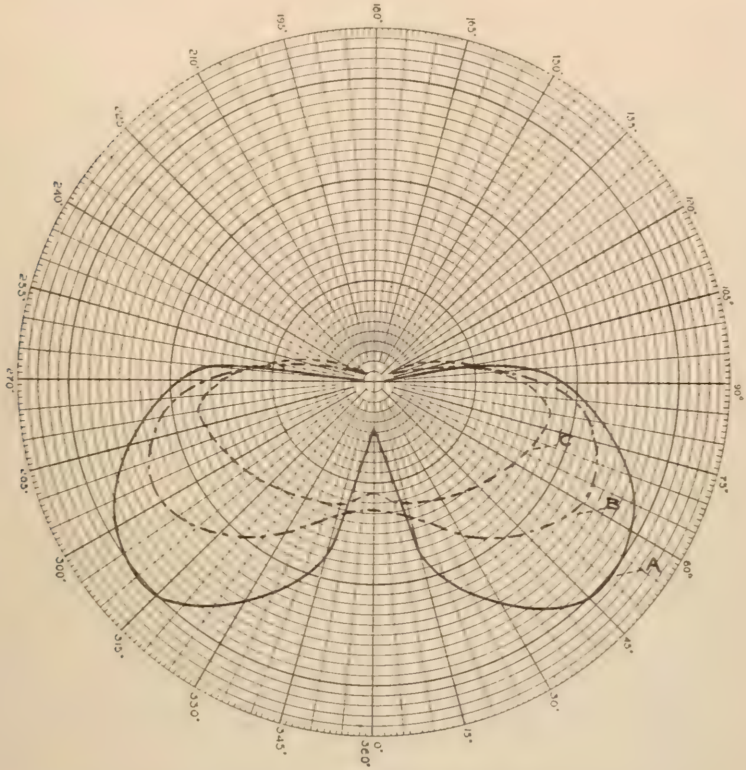

FIG. 11. EFFECT OF GLOBE

A. Vertical Electrode, Flame Arc Lamp with Reflector and Clear Outer Globe.
B. Same Lamp with Alba Globe.
C. Same Lamp with Dense Opal Globe.

V. CALCULATION OF ILLUMINATION

12 *General Theory.*—In order to obtain the best results from general illumination, it should be as nearly uniform as possible over the whole surface. The object of street illumination is to make the street safer for travel after nightfall. To accomplish this result, the eye must be able to distinguish objects with sufficient clearness and at a fair distance, so that they may be avoided if necessary. With the advent of the automobile and other swiftly moving vehicles, the distinctness with which objects may be seen must be improved, so that danger may easily be avoided.

The illumination produced by full moon has a maximum of about .02 foot candle. The eye has become accustomed to distinguish objects fairly well by this illumination, even at a considerable distance. Moreover, this illumination is uniform except where some object, such as a tree or a building, intervenes to shut off the light. This illumination is not sufficient for reading ordinary type without fatigue. For reading purposes from 1 to 3 foot candles are required. The eye is accustomed to adjust itself to quite a wide range of intensity, if the changes in intensity do not follow each other too rapidly. A street lighted by 6.6 amperes alternating current enclosed arc lights placed at the street crossings 25 ft. above the street surface where blocks are 400 ft. long, has a variation in its horizontal illumination of from .28 foot candle, near the lamp, to .0014 foot candle midway between the lamps. With the lamps placed at alternate street intersections, as is the custom in smaller cities, the minimum illumination, i. e. the illumination near the other street intersections, will be less than .0001 foot candle, an almost negligible quantity. An automobile traveling at a speed of 25 miles per hour along a street lighted every 400 ft. will pass from the point of minimum to maximum illumination in about 5.5 seconds. During this time, the eye must accustom itself to a range in illumination of 200 to 1, producing fatigue in the eye and uncertainty in discerning objects. When the illumination is not uniform, but varies over a wide range from lamp to lamp, the eye is not able to distinguish objects nearly so well, at the points of minimum illumination, as it is when the illumination is more nearly uniform but low.

The intrinsic brilliancy of the light source in the line of vision is another consideration in selecting the proper lamps, as well as their height and spacing. The pupil of the eye adjusts itself so as to admit a certain quantity of light. A source of light of high intrinsic brilliancy, in the line of vision, closes the pupil of the eye so that insufficient light is admitted to distinguish dimly lighted objects clearly. This effect is made apparent, in ordinary street lighting, by the fact that a person facing an arc light is unable to see objects beyond the light until he is so near the light that it is no longer in his line of vision. The effect is not so apparent in brightly lighted streets where the illumination is also more uniform.

13. *Tables and Curves.*—(*a*) In order to calculate the illumination on the street surface, the following equations have been

derived. Fig. 12 represents a light, L, suspended at a height, h, above the center of the street, BC. The intensity of the light in the direction, LB, at an angle ϕ with the vertical may be

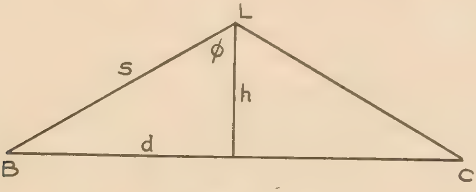

FIG. 12

obtained from the distribution curve of the lamp. Let this intensity of the light source be I. Then the intensity of the illumination at B, on a plane perpendicular to the ray of light, would be

$$i = \frac{I}{s^2} \qquad (1)$$

The illumination on the horizontal surface of the street would be

$$i_h = \frac{I \cos \phi}{s^2} \qquad (2)$$

since the ray is at the angle ϕ with the vertical. Taking the point B, at a horizontal distance d, from the lamp

$$\tan \phi = d/h \qquad (3)$$

and

$$s = h/\cos \phi \qquad (4)$$

Hence the normal illumination, or the illumination on a plane perpendicular to the ray of light would be

$$i = \frac{I \cos^2 \phi}{h^2} \qquad (5)$$

and on the street surface

$$i = \frac{I \cos^3 \phi}{h^2} \qquad (6)$$

In order to have uniform normal illumination, the intensity in any direction about the lamp must vary according to the equation

$$I = \frac{i\, h^2}{\cos^2 \phi} \qquad (7)$$

derived from equation (5) or

$$I = \frac{i\, h^2}{\cos^3 \phi} \qquad (8)$$

from equation (6), for uniform horizontal distribution. In these two equations, if

h = height of lamp in feet above the street;

and

I = intensity of the light in candle power at the angle ϕ from the vertical;

then

i = intensity of illumination in foot candles.

Table 1 has been calculated from equation (7) for different values of h, ranging from 15 to 50 ft. In the same manner, Table 2 has been calculated from equation (8). In each table, a uniform intensity of illumination of .01 foot candle has been assumed. For any other intensity, these values should be

TABLE 1

CANDLE POWER AT LAMP FOR .01 FOOT CANDLE NORMAL ILLUMINATION ON STREET.

ϕ	$\cos\phi$	$\frac{1}{\cos^2\phi}$	Height of lamp—feet					
			15	20	25	30	40	50
0	1.000	1	2.25	4.00	6.25	9.00	16.00	25.00
5	.996	1.01	2.27	4.04	6.31	9.09	16.16	25.25
10	.985	1.03	2.32	4.12	6.44	9.27	16.48	25.75
15	.966	1.07	2.41	4.28	6.69	9.63	17.12	26.75
20	.940	1.13	2.57	4.56	7.13	10.17	18.26	28.50
25	.906	1.22	2.75	4.88	7.63	10.98	19.52	30.50
30	.866	1.33	2.99	5.32	8.31	11.97	21.30	33.30
35	.819	1.49	3.35	5.96	9.32	13.41	23.80	37.3
40	.766	1.70	3.83	6.80	10.60	15.3	27.2	42.5
45	.707	2.00	4.50	8.00	12.50	18.0	32.0	50.0
50	.643	2.43	5.47	9.72	15.16	21.8	38.9	60.8
55	.547	3.03	6.33	12.12	18.93	27.3	48.5	76.0
60	.500	4.00	9.00	16.0	25.0	36 0	64.0	100.0
65	.423	5.59	12.68	22.4	35.0	50.3	89.7	140.0
67.5	.383	6.82	15.35	27.3	42.6	61.4	109	170.5
70	.342	8.35	18.8	33.4	52.5	75.1	133.5	209
72.5	.301	11.05	24.9	44.2	69.1	99.5	177	276
75	.259	14.90	34.0	60.4	94.4	136	242	378
76	.242	17.10	38.5	68.4	107.	154	274	428
77	.225	19.75	44.5	79.0	123	178	316	494
78	.208	23.10	52.0	92.4	145	209	370	577
79	.191	27.4	61.7	109.6	171	246	438	687
80	.1736	33.2	74.8	133.	208	299	532	832
81	.1564	40.8	92.2	163	255	367	653	1020
82	.1392	51.7	116.	207	323	465	830	1292
83	.1219	67.3	151	269	421	606	1077	1682
84	.1045	91.7	206	367	573	826	1467	2290
85	.0872	131.7	296	527	824	1180	2110	3290
85.5	.0785	162.5	366	650	1010	1460	2600	4060
86.	.0698	205	461	820	1300	1850	3280	5120
86.5	.0610	269	605	1070	1680	2420	4300	6730
87.	.0523	366	824	1460	2290	3300	5850	9150
87.25	.0480	434	980	1740	2710	3900	6950	10850
87.5	.0436	527	1185	2110	3300	4740	8450	13170
87.75	.03926	650	1460	2600	4060	5850	10400	16250
88.	.03490	822	1850	3290	5140	7400	13150	20550
88.25	.03054	1072	2300	4290	6700	9540	17150	26800
88.5	.02618	1460	3290	5840	9120	13100	23400	36500
89.	.01745	3290	7420	13100	20500	29600	52700	82400

multiplied by the ratio between .01 and the intensity chosen. Thus, for .10 foot candle, the values should be multiplied by 10.

TABLE 2

CANDLE POWER AT LAMP FOR .01 FOOT CANDLE HORIZONTAL ILLUMINATION AT STREET

ϕ	$\cos\phi$	$\frac{1}{\cos^3\varphi}$	Height of Lamp					
			15′	20′	25′	30′	40′	50′
0	1.000	1.000	2.25	4.00	6.25	9.00	16.00	25.0
5	.996	1.015	2.28	4.06	6.35	9.14	16.25	25.4
10	.985	1.045	2.35	4.18	6.43	9.41	16.72	26.15
15	.966	1.11	2.50	4.44	6.94	9.99	17.76	27.75
20	.940	1.20	2.70	4.80	7.50	10.80	19.20	30.0
25	.906	1.35	3.04	5.00	8.44	12.15	21.60	33.7
30	.866	1.54	3.47	6.16	9.53	13.86	24.65	38.5
35	.819	1.82	4.09	7.28	11.35	16.4	29.1	45.5
40	.766	2.22	5.00	8.88	13.85	20.0	35.5	55.2
45	.707	2.83	6.37	11.30	17.68	25.4	45.3	70.8
50	.643	3.73	8.42	14.9	23.3	33.6	59.7	93.5
55	.547	5.27	11.85	21.1	32.9	47.4	84.5	131.7
60	.500	8.00	18.00	32.0	50.0	72.0	128.0	200
65	.423	13.20	29.7	53.8	82.5	119	211	330
67.5	.383	17.8	40.1	71.2	111	160	285	445
70	.342	24.4	54.9	97.6	152	220	390	610
72.5	.301	36.7	82.6	146	229	330	587	918
75	.259	58.3	131.2	233	364	525	935	1460
76	.242	70.5	159	282	440	635	1130	1760
77	.225	87.8	198	351	549	790	1410	2200
78	.208	111.2	250	445	695	1000	1780	2780
79	.191	143.3	323	574	896	1290	2290	3590
80	.1736	191.0	430	764	1190	1720	3060	4780
81	.1564	261.0	574	1040	1630	2350	4180	6530
82	.1392	371.0	835	1480	2320	3340	5940	9280
83	.1219	552.0	1242	2210	3450	4970	8850	13800
84	.1045	876.0	1970	3500	5480	7880	14000	21900
85	.0872	1510	3400	6040	9440	13600	24200	37800
85.5	.0785	2070	4660	8280	12900	18600	33100	51800
86	.0698	2930	6600	11720	18320	26380	46800	73400
86.5	.0610	4410	9950	17650	27600	39700	70600	110000
87	.0523	6970	15700	27900	43500	62700	111500	174200
87.25	.0480	9050	20400	36200	56600	81500	145000	226000
87.5	.0436	12100	27200	48400	75600	118000	194000	303000
87.75	.03926	16500	37100	66000	103000	148000	264000	413000
88	.03490	23500	52900	94000	146500	211000	376000	588000
88.25	.03054							
88.5	.02618							
89	.01745							

From equation (3), the value of ϕ has been calculated for different values of d and h over as wide a range as necessary for ordinary exterior illumination. These angles are given in Table 3. Factors have also been included in Table 3 to aid in the calculation of illumination. For each height and distance, two factors are given by which the intensities in thousands of candle power must be multiplied to give the illumination at the chosen point. The first line is for normal and the second for horizontal illumination, respectively. By referring to Tables 1 and 2, the difficulty will be seen of securing uniform low illumination over large areas with economical height and spacing of lamps. Higher values of intensity may be secured by placing the lamps sufficiently close so that the illumination from one assists that from the others, increasing the minimum illumination.

TABLE 3

TABLE OF ANGLES AND FACTORS FOR VARIOUS HEIGHTS AND DISTANCES

$$a = \frac{\cos^2 \phi}{h^2} \times 1000 \text{—Normal illumination}$$

$$b = \frac{\cos^3 \phi}{h^2} \times 1000 \text{—Horizontal illumination}$$

Distance feet	Height—feet					
	15	20	25	30	40	50
25	59°2′	51°20′	45°-0′	39°50′	31°-0′	26°-3″
a	1.18	.977	.800	.656	.458	.321
b	.607	.610	.566	.504	.393	.287
50	73°18′	68°12′	63°27′	59°2′	51°20′	45°0′
a	.366	.345	.323	.295	.244	.200
b	.105	.128	.143	.150	.152	.141
75	78°41′	75°4′	71°34′	68°12′	62°0′	56°20′
a	.171	.168	.161	.153	.138	.123
b	.0335	.043	.051	.057	.065	.0682
100	81°28′	78°41′	75°57′	73°20′	68°12′	63°26′
a	.100	.0987	.0915	.0956	.0863	.0804
b	.0148	.0193	.022	.027	.032	.036
125	83°10′	80°55′	78°41′	76°30′	72°20′	68°10′
a	.0662	.0625	.0615	.0607	.0577	.0553
b	.0079	.010	.012	.0141	.0125	.0205
150	84°17′	82°24′	80°32′	78°41′	75°4′	71°34′
a	.0452	.0436	.0433	.0427	.0416	.0400
b	.0045	.0058	.0071	.0084	.0107	.0126
200	85°42′	84°17′	82°52′	81°28′	78°41′	75°57′
a	.0254	.0252	.0247	.0245	.0234	.0233
b	.0019	.0025	.0031	.0036	.0046	.0056
250	86°34′	85°25′	84°17′	83°10′	80°55′	78°41′
a	.016	.016	.016	.0158	.0156	.0154
b	.00106	.00128	.0016	.0019	.00247	.0030
300	87°8′	86°11′	85°14′	84°17′	82°24′	80°32′
a	.0111	.0111	.0111	.011	.0109	.0108
b	.00055	.00173	.00091	.0011	.00144	.0018
350	87°33′	86°44′	85°50′	85°5′	83°29′	81°57′
a	.00812	.00812	.00812	.00812	.00807	.0077
b	.00035	.00046	.00058	.00070	.00091	.0011
400	87°51′	87°8′	86°25′	85°42′	84°17′	82°52′
a	.00625	.00625	.00625	.00625	.0061	.0060
b	.00023	.00031	.00039	.00047	.0006	.00074
450	88°5′	87°27′	86°49′	86°11′	84°55′	83°40′
a	.0050	.00494	.00492	.00491	.00490	.00487
b	.00017	.00022	.00027	.00032	.00043	.000536
500	88°17′	87°42′	87°8′	86°34′	85°25′	84°17′
a	.004	.004	.004	.004	.004	.004
b	.00012	.00016	.00020	.00024	.00032	.0004

This method is used in interior illumination, and in the brilliant illumination of streets. From Table 3 and the distribution curves of the several lamps, Tables 4 to 13 have been computed. These tables give the foot candles illumination on the normal and horizontal planes for different heights and spacing of the lamps.

TABLE 4

Normal and Horizontal Distribution at Street Surface for 9.6 Amperes D. C. Open Arc Lamp

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	.222	1.185	.242	.089	.0476	.030	.019	.0104	.0065	.0044	.0032	.0025
	.222	.610	.0693	.0174	.0070	.00355	.0019	.00078	.00043	.00028	.00014	.00009
20	.125	1.18	.228	.101	.050	.031	.021	.011	.0067	.0044	.0031	.0024
	.125	.735	.0845	.0258	.0098	.0049	.0028	.0011	.00054	.00029	.00017	.00012
25	.080	1.00	.294	.120	.057	.031	.022	.011	.0064	.0046	.0038	.0025
	.080	.700	.133	.0384	.0130	.0061	.0035	.0014	.00068	.00038	.00024	.00016
30	.0555	.788	.303	.116	.063	.036	.022	.012	.0068	.0046	.0034	.0026
	.0555	.605	.154	.0435	.018	.0083	.0043	.0018	.00082	.00046	.0003	.00019
40	.0312	.417	.296	.138	.069	.038	.025	.012	.0076	.0050	.0034	.0026
	.0312	.356	.185	.065	.026	.012	.0064	.0023	.0012	.00066	.00038	0 0025
50	.020	.284	.250	.140	.0735	.044	.028	.0137	.0079	.0052	.0037	0026
	.020	.253	.176	.0775	.033	.0165	.0089	.0033	.0015	.00089	.00052	.00032

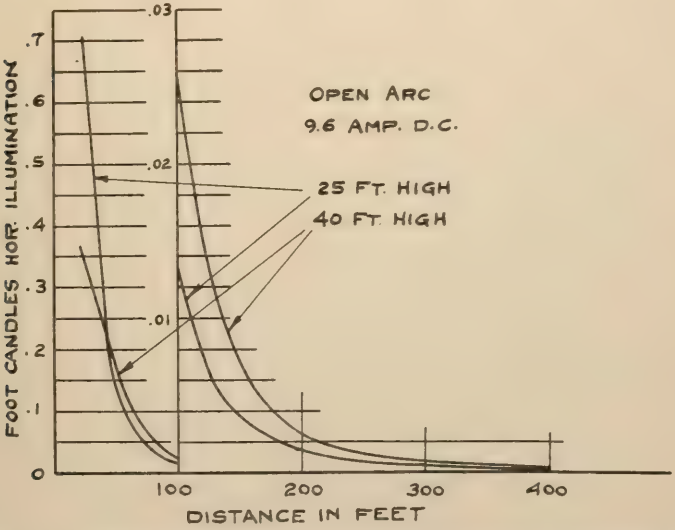

Fig. 13. Horizontal Distribution at Street Surface for 9.6 Amp. D. C. Open Arc Lamp

TABLE 5

Normal and Horizontal Distribution at Street Surface for 6.6 Amperes D. C. Magnetite Arc Lamp

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	2.22	1.76	.550	.269	.155	.100	.0660	.0320	.0208	.0142	.0101	.0075
	2.22	.91	.169	.053	.023	.0118	.0066	.0024	.00138	.00070	.00044	.00028
20	1.25	1.30	.506	.456	.134	.0950	.0658	.0357	.0223	.0146	.0104	.0080
	1.25	.815	.188	.0656	.0267	.0153	.0088	.0035	.00178	.00096	.00059	.0004
25	.60	.985	.480	.242	.142	.0970	.0675	.0366	.0236	.0154	.0109	.0083
	.60	.696	.225	.0765	.0342	.0189	.0111	.0046	.0024	.0013	.00078	.00051
30	.556	.590	.437	.225	.143	.0937	.0635	.0387	.0235	.0160	.0114	.0085
	.556	.454	.223	.0840	.0405	.0218	.0125	.0056	.0028	.0016	.0010	.00064
40	.313	.435	.330	.203	.126	.0872	.0635	.0370	.0246	.0168	.0119	.0086
	.313	.373	.205	.0955	.0470	.0265	.0163	.00727	.0039	.0022	.00134	.00084
50	.200	.257	.246	.178	.117	.0813	.0595	.0356	.0243	.0171	.0116	.0090
	.200	.230	.173	.099	.052	.0301	.0188	.00856	.0047	.00286	.0017	.00111

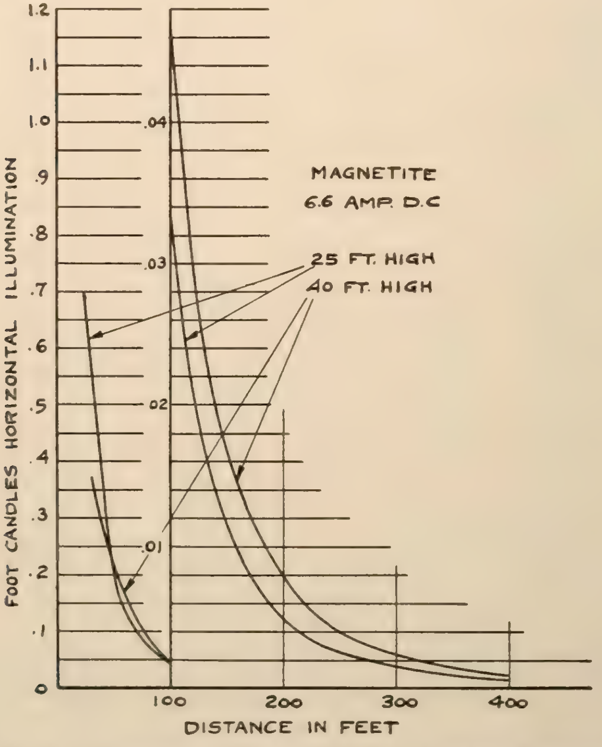

Fig. 14. Horizontal Distribution at Street Surface for a 6.6 Amp. Magnetite Arc Lamp

TABLE 6

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 4 AMPERES D. C. MAGNETITE ARC LAMP

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	1.55	.768	.259	.125	.0742	.0459	.0303	0155	.01015	.0069	.0050	.00375
	1.55	.395	0743	.0244	.0110	.00547	.00302	.00116	.00067	.00034	.00021	.00014
20	.875	.596	.237	.1210	.0713	.0454	.0305	.0162	.0107	.0071	.0050	.0039
	.875	.372	.0872	.0310	.0140	00726	.00406	.00161	.00086	.00047	.00028	.00019
25	.560	.440	.213	.113	.066	.0448	.0311	.0163	.0110	.00743	.00540	.0040
	.560	.311	.095	.0357	.0159	.00875	.0051	.00217	.0011	.00061	.00037	.00025
30	.389	.328	.192	.098	.0677	.0438	.0311	.0167	.0110	.00765	.00543	,00445
	.389	.252	.0975	.0366	.0191	.0102	.00613	.0026	.0013	.00076	.00047	.0003
40	.219	.167	.148	.091	.0588	.0400	.0300	.0160	.0112	.0076	.0057	.0040
	.219	.143	.092	.0428	.0218	.0123	.0077	.00334	.00177	.0010	.00063	.0004
50	.140	.108	.110	.0718	.0537	.0380	.0278	.0158	.0112	.0079	.0055	.0042
	.140	.0965	.0777	.0398	.0241	.0141	.00875	.0040	.0022	.0013	.00078	.0005

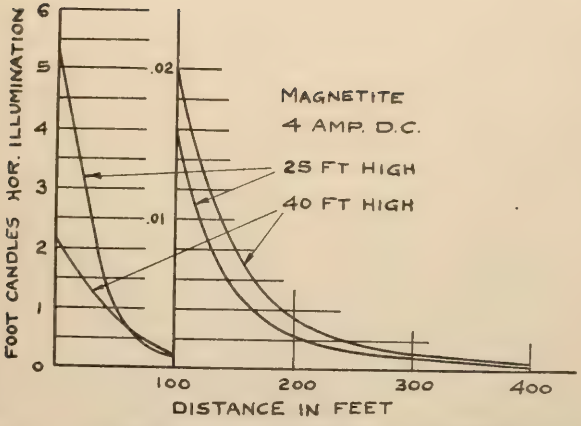

FIG. 15. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 4.0 AMP. MAGNETITE ARC LAMP

TABLE 7

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 6.6 AMPERES D. C. ENCLOSED ARC LAMP

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	1.00	.62	.187	.083	.0475	.0310	.0183	.0113	.0075	.00485	.00355	.0027
	1.00	.32	.054	.012	.007	.0036	.0018	.00045	.00036	.00024	.00015	.0001
20	.563	.503	.180	.084	.05	.0296	.0202	.0115	.0072	.0049	.0035	.0027
	.563	.314	.067	.022	.0097	.0048	.0027	.0011	.0006	.0003	.0002	.0001
25	.36	.400	.167	.083	.046	.0296	.0206	.0116	.0073	.0050	.0036	.0028
	.36	.283	.075	.026	.011	.0058	.0034	.0014	.00072	.00041	.00026	.00017
30	.25	.311	.155	.0803	.0490	.0304	.0207	.0116	.0074	.0050	.00366	.0028
	.25	.240	.089	.0298	.0138	.0071	.0041	.0017	.00089	.00050	.00031	.00021
40	.14	.197	.127	.0725	.0453	.0297	.0208	.0112	.0074	.00518	.0037	.0027
	.14	.167	.079	.0341	.0168	.0030	.0054	.0022	.0012	.00067	.00042	.00027
50	.091	.128	.100	.0645	.0420	.0290	.0207	.0117	.0075	.00512	.00364	.0028
	.091	.115	.072	.0356	.0189	.0108	.0065	.0050	.00146	.00085	.00052	.00034

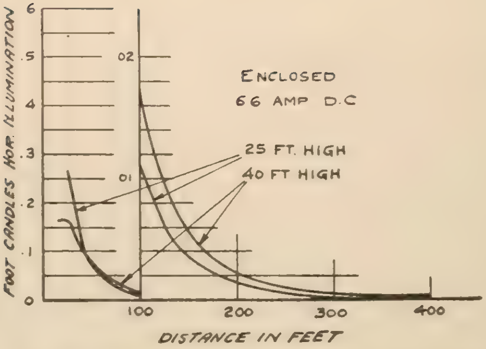

FIG. 16. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 6.6 AMP. D. C. ENCLOSED ARC LAMP

TABLE 8

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 6.6 AMPERES A. C. ENCLOSED ARC LAMP

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	.78	.313	.086	.036	.021	.014	.0097	.0053	.0033	.0022	.0016	.0012
	.78	.161	.0247	.0075	.0032	.0017	.00096	.0004	.0002			
20	.44	.259	.077	.038	.022	.014	.0094	.0053	.0034	.0023	.0017	.0013
	.44	.168	.0285	.0097	0043	.0022	.0012	.0005	.00027			
25	.28	.22	.080	.038	.022	.014	.0095	.0054	.0034	.0024	.0017	.0013
	.28	.156	.032	.012	.0044	.0027	.00156	.0007	.00034			
30	.19	.18	.071	038	.022	.0137	.0094	.0054	.0034	.0023	.0017	.0013
	.19	.139	.0362	.014	.0063	.032	.0018	.0008	.0004	.0002		
40	.11	.124	.067	.036	.022	.0135	.0095	.0054	.0034	.0024	.0017	.0013
	.11	.116	.042	.017	.008	.0041	.0024	.0011	.0005	.0003		
50	.07	.083	.055	.033	.020	.0135	.010	.005	.0034	.0024	.0016	.0013
	.07	.074	.039	.018	.0092	.005	.003	.0022	.0006	.0004	.0002	.0001

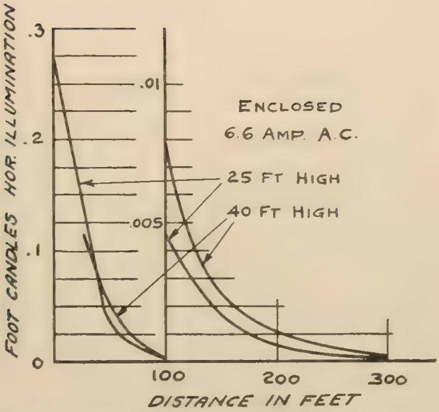

FIG. 17. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 6.6 AMP. A. C. ENCLOSED ARC LAMP

TABLE 9

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 7.5 AMPERES A. C. ENCLOSED ARC LAMP

Height feet	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	.94	.376	.103	.043	.025	.017	.011	.0063	.004	.0022	.0019	.0014
	.94	.193	.030	.009	.0038	.002	.001	.0005	.0002			
20	.53	.311	.092	.045	.026	.017	.011	.0064	.0041	.0027	.0020	.0015
	.53	.201	.034	.0016	.0051	.0026	.0014	.0006	.0003			
25	.336	.26	.096	.045	,026	.017	011	,0065	.0041	.0029	.0020	.0015
	.336	.187	.038	.014	.0053	.0032	.0019	.0008	.0004			
30	.228	.216	.085	.045	.026	.0165	.011	.0065	.004	.0028	.002	.0015
	.228	.167	.043	.017	.0076	.0038	.0022	.0011	.0005	.0002		
40	.132	.149	.080	.043	.026	.0162	.011	.0066	.0041	.0029	.0020	.0015
	.132	.139	.050	.020	.0096	.0049	.0029	.0013	.0006	.0004		
50	.084	.10	.066	.040	.024	.016	.012	.006	.0041	.0029	.0019	.0015
	.084	.09	.047	.022	.011	.006	.0036	.0026	.0007	.0005	.0002	.0001

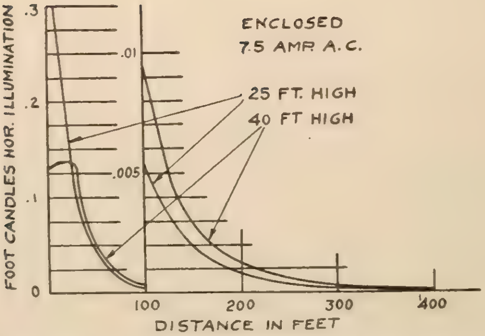

FIG. 18. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 7.5 AMP. A. C. ENCLOSED ARC LAMP

TABLE 10

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 10 AMPERES D. C. FLAME ARC LAMP, INCLINED ELECTRODES, OPAL GLOBES

Height feet	Distance											
	0	25	50	75	100	125	150	200	250	300	350	400
15	10.6	2.33	.585	.240	.142	.085	.056	.031	.019	.013	.010	.007
	10.6	1.20	.168	.047	.021	.010	.0055	.0022	.0013	.0006	.0004	.0003
20	5.95	1.70	.595	.259	.136	.084	.057	.031	.020	.013	.010	.008
	5.95	1.28	.224	.066	.027	.0135	.0075	.0031	.0015	.0008	.0005	.0004
25	3.81	1.77	.602	.262	.140	.085	.058	031	.020	.013	.010	.0075
	3.81	1.25	.265	.084	.034	.017	.010	.004	.002	.0011	.0007	.0005
30	2.65	1.51	.580	.266	.152	.091	.059	.032	.020	.014	.010	.0075
	2.65	1.15	.297	.100	.043	.021	.012	.0047	.0025	.0014	.0008	.0075
40	1.49	1.09	.52	.253	.150	.094	.064	.032	.021	.014	.010	.0077
	1.49	.94	.32	.123	.055	.029	.0165	.0064	.0032	.0018	.0011	.0008
50	.95	.76	.44	.25	.16	.096	.065	.035	.022	.015	.010	.0084
	.95	.68	.315	.138	.072	.036	.021	.0084	.0042	.0024	.0014	.001

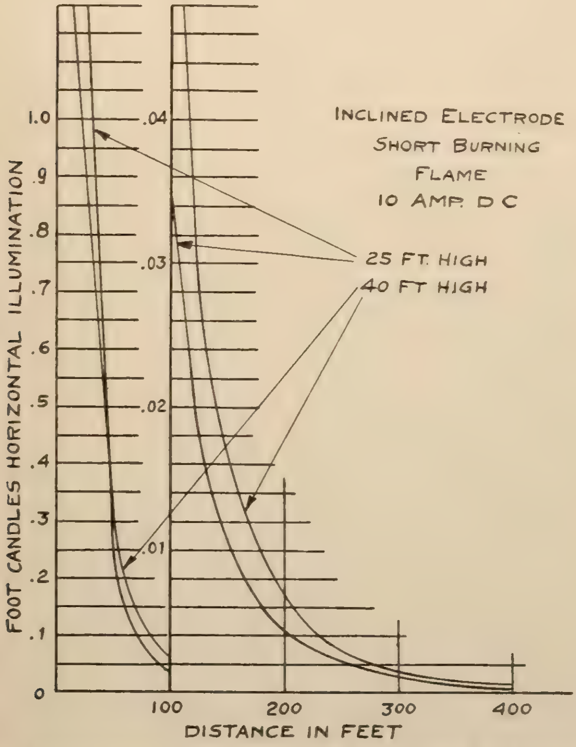

FIG. 19. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 10 AMP. D. C. FLAME ARC LAMP, INCLINED ELECTRODES

TABLE 11

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 5.5 AMPERES D. C. LONG BURNING FLAME ARC LAMP VERTICAL ELECTRODES, OPAL GLOBES

	Distance—feet											
	0	25	50	75	100	125	150	200	250	300	350	400
15	3.38	1.77	.65	.32	.19	.125	.087	.049	.0304	.021	.015	.011
	3.38	.91	.186	.063	.0281	.015	.0087	.0036	.0020	.001	.0006	.0004
20	1.90	1.49	.587	.302	.182	.119	.082	.0485	.031	.021	.0154	.0115
	1.90	.945	.218	.0775	.036	.019	.011	.0048	.0025	.0033	.0009	.0006
25	1.26	1.0	.550	.278	.166	.114	.081	.047	.031	.021	.0156	.012
	1.26	.707	.243	.088	.040	.022	.0132	.0059	.0031	.0018	.0011	.0007
30	.845	.760	.443	.260	.166	.110	.079	.046	.030	.021	.0156	.012
	.845	.657	.225	.095	.047	.0256	.0155	.0068	.0036	.0021	.0013	.0009
40	.475	.458	.398	.214	.146	.101	.075	.043	.03	.021	.015	.0116
	.475	.393	.204	.101	.054	.022	.0193	.0085	.0047	.0027	.0017	.0011
50	.304	.321	.25	.180	.128	.094	.069	.042	.029	.020	.015	.0114
	.304	.287	.176	.10	.057	.035	.0217	.010	.0056	.0034	.0021	.0014

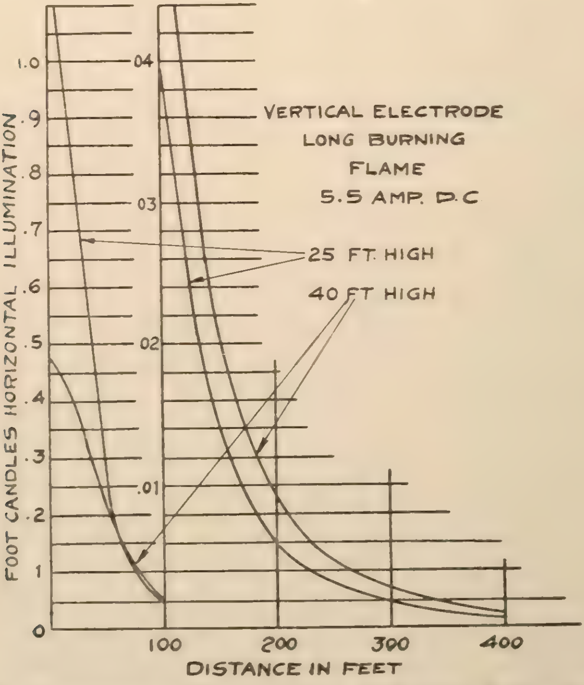

FIG. 20. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 5.5 AMP. D. C. LONG BURNING FLAME ARC LAMP VERTICAL ELECTRODES

TABLE 12

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 200 C. P. TUNGSTEN WITH RADIAL WAVE REFLECTOR

Height feet	Distance—feet							
	0	25	50	75	100	125	150	200
15	.643	.316	.097	.043	.026	.015	.0102	.0057
	.643	.163	.028	.0084	.0039	.0018	.0010	.0004
20	.362	.243	.093	.044	.025	.0153	.0105	.0058
	.362	.155	.035	.0112	.0049	.0024	.0014	.0006
25	.232	.200	.086	.0427	.0238	.0154	.0107	.0059
	.232	.141	.035	.0135	.0057	.0030	.0017	.0007
30	.161	.154	.078	.0413	.0251	.0156	.0107	.0060
	.161	.118	.040	.0154	.0071	.0036	.0021	.0009
40	.091	.098	.063	.0376	.0233	.0153	.0108	.006
	.091	.084	.039	.0175	.0086	.0046	.0028	.0011
50	.058	.062	.049	.0326	.0217	.0151	.0106	.006
	.058	.058	.035	.018	.0097	.0054	.0033	.0014

TABLE 13

NORMAL AND HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR 350 C. P. TUNGSTEN WITH FLUTED ENAMELED REFLECTOR

Height feet	Distance—feet									
	0	25	50	75	100	125	150	200	250	300
15	1.03	.503	.165	.072	.041	.0265	.0179	.010	.0062	.0043
	1.03	.259	.0473	.014	.006	.0031	.0018	.00074	.00041	.00021
20	.575	.395	.156	.074	.042	.0256	.0178	.010	.0063	.0043
	.575	.247	.0575	.019	.0082	.0041	.0024	.001	.0005	.00024
25	.370	.324	.141	.0725	.040	.0252	.0180	.017	.0063	.00435
	.370	.230	.063	.023	.0095	.0045	.0030	.0012	.0006	.00036
30	.257	.256	.126	.0685	.0425	.0262	.018	.010	.0063	.0044
	.257	.197	.064	.026	.0120	.0061	.0035	.0015	.00076	.00044
40	.145	.168	.099	.060	.0392	.0259	.0181	.0098	.0064	.0044
	.145	.145	.0615	.0383	.0145	.0079	.0046	.0019	.0010	.00058
50	.093	.111	.081	.052	.0354	.025	.0181	.010	.0065	.0045
	.093	.099	.057	.0288	.0160	.0093	.0057	.0024	.0013	.00074

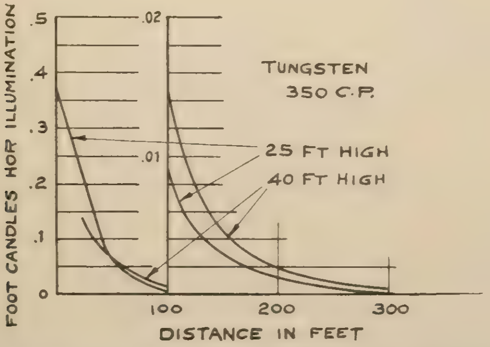

FIG. 21. HORIZONTAL DISTRIBUTION AT STREET SURFACE FOR A 350 C. P. TUNGSTEN LAMP WITH ENAMELLED REFLECTOR

(*b*) *Surface Illumination Curves.*—From the data in Tables 4 to 13, curves have been plotted, showing the distribution of illumination over the street surface for each of the lamps. Two curves are plotted in each case, one for a height of 25 ft. and the other for 40 ft. At a distance of 100 ft. from the base of the lamp, the illumination has become so low that it is not easily comparable with that near the lamp. In order to read these values

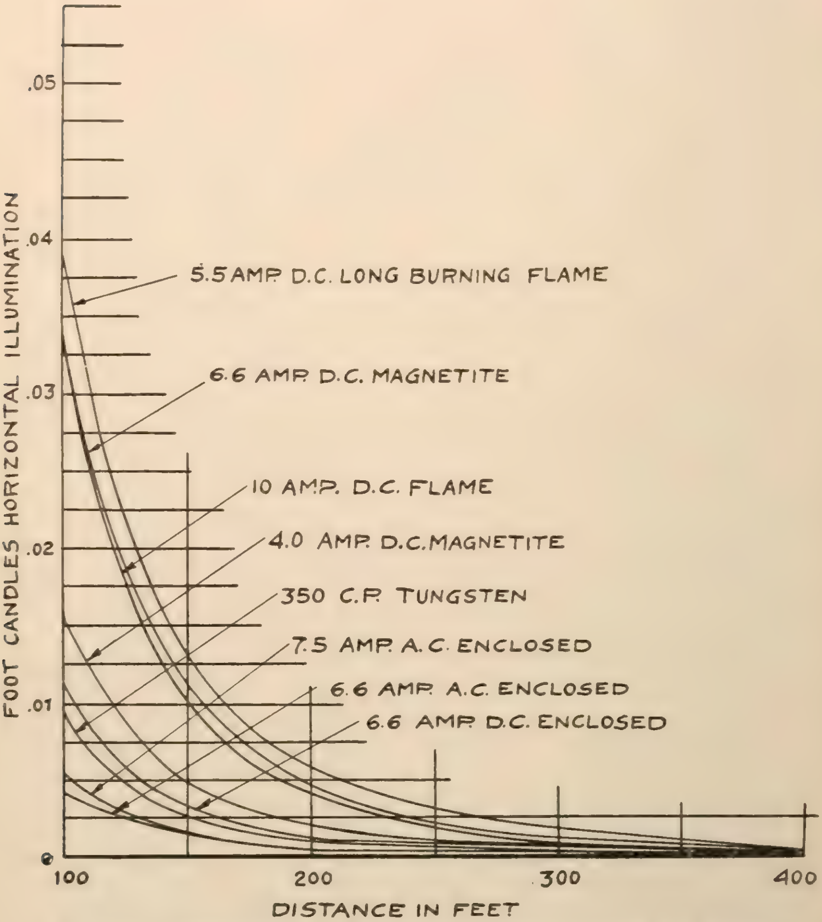

FIG. 22. COMPARISON OF HORIZONTAL DISTRIBUTION AT STREET SURFACE BY DIFFERENT LAMPS

more easily, the foot candle scale has been magnified between 100 and 400 ft. These curves are shown in Fig. 13 to 21. In Fig. 22, a comparison is made between the different lamps between 100 and 400 ft.

In order to indicate the method of computing the distribution of illumination on the street surface due to several lamps, and the effect of height and spacing, the following calculations have been made. Take, as an example, the 6.6 ampere alternating current enclosed arc, suspended 25 and 40 ft. above the center of the street surface, and spaced 800, 400 and 200 ft. apart, these distances being controlled by local conditions. In Fig. 23, OO′ represents the surface of the street and 1-2 lamps. It is obvious that the normal illumination can not be added for lamps on opposite sides of an object, but that the horizontal is increased. Adding the illumination from the two lights at the various distances, line 1, Table 14 is obtained from Table 8. It is seen that the illumination beyond 400 ft. from the lights is so small that the sum need be considered only at the middle point.

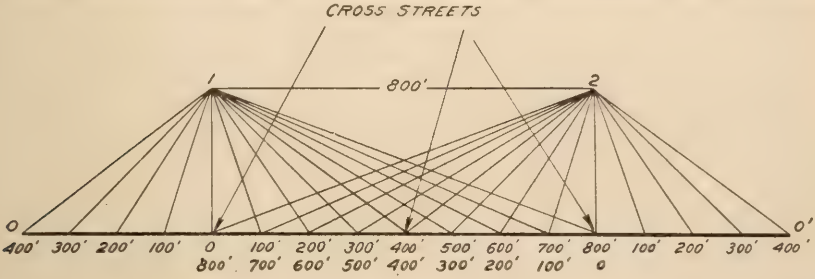

FIG. 23

For 400 ft. spacing, line 2 is obtained from the two lights 1-2 at street intersections (see Fig. 24). It will be seen that the illumination at any point is that produced by the sum of the illuminations from all lights at their respective distances from that point. Only two lights need be considered at 400 ft. At 200 ft. it will be found that four lights must be considered. In each

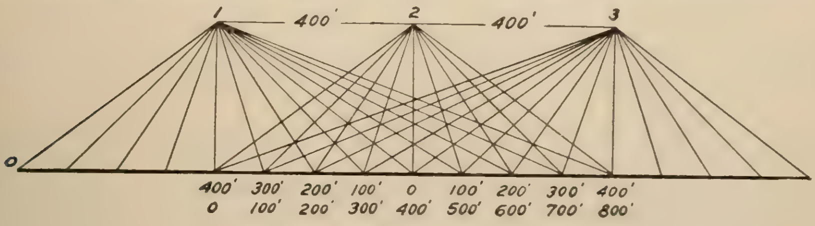

FIG. 24

case, the maximum illumination is affected but little, but the general effect becomes better. For 200 ft. spacing, line 3, Table 14 is computed. In a similar manner, the illumination for other types of lights has been computed. These are also illustrated in Table 14. These values have not been plotted in the form of curves.

TABLE 14

Horizontal Distribution at Street Surface for Different Heights and Spacings

	Spacing feet	h feet	Distance—feet								
			0	50	100	150	200	250	300	350	400
6.6 Amp. D. C. Enclosed	800	25	.36	.075	.011	.0034	.0014	.0007	.0004	.0004	.0004
	400	25	.36	.075	.011	.04	.003	.004	.011	.075	.36
	200	25	.372	.082	.023	.082	.372	.082	.023	.082	.372
	800	40	.14	.079	.0168	.0054	.0022	.0012	.0007	.0006	.0006
	400	40	.14	.08	.017	.0066	.0044	.0066	.017	.08	.14
	200	40	.142	.085	.034	.085	.142	.085	.034	.085	.142
7.5 Amp. A. C. Enclosed	800	25	.336	.038	.005	.0019	.0008	.0004	.0003	.0002	.0001
	400	25	.336	.038	.0053	.0023	.0016	.0023	.0053	.038	.336
	200	25	.34	.05	.012	.05	.34	.05	.012	.05	.34
	800	40	.132	.05	.0096	.003	.0013	.0006	.0004	.0003	.0002
	400	40	.132	.05	.011	.0035	.0026	.0035	.011	.05	.132
	200	40	.135	.10	.022	.10	.135	.10	.022	.10	.135
6.6 Amp. D. C. Magnetite	800	25	.80	.225	.034	.0111	.0046	.0024	.0013	.001	.001
	400	35	.80	.226	.035	.0135	.0092	.0135	.035	.226	.80
	200	25	.85	.238	.069	.238	.85	.238	.069	.235	.85
	800	40	.315	.205	.047	.0163	.0073	.0039	.0022	.002	.0016
	400	40	.316	.206	.049	.020	.014	.020	.049	.206	.316
	200	40	.329	.225	.096	.225	.329	.225	.096	.225	.329
4.0 Amp. D. C. Magnetite	800	25	.56	.095	.016	.005	.002	.001	.0016	.0006	.0005
	400	25	.56	.095	.016	.006	.004	.006	.016	.095	.56
	200	25	.564	.101	.033	.101	.564	.101	.033	.101	.564
	800	40	.219	.092	.022	.0077	.0033	.0018	.001	.0009	.0008
	400	40	.219	.092	.044	.0095	.0066	.0095	.044	.092	.219
	200	40	.222	.094	.044	.094	.222	.094	.044	.094	.222
9.6 Amp. D. C. Open	800	25	.08	.133	.013	.0035	.0014	.0007	.0004	.0003	.0003
	400	25	.08	.133	.013	.042	.0028	.042	.013	.133	.08
	200	25	.081	.136	.026	.136	.081	.136	.026	.136	.081
	800	40	.031	.185	.026	.006	.002	.001	.0007	.0006	.0005
	400	40	.031	.185	.026	.007	.004	.007	.026	.185	.031
	200	40	.035	.192	.053	.192	.035	.192	.053	.192	.035

TABLE 14 (*Continued*)

	Spacing feet	h feet	Distance—feet								
			0	50	100	150	200	250	300	350	400
Long Burning Flame Vertical	800	25	1.26	.243	.040	.0132	.0059	.0031	.002	.0016	.0014
	400	25	1.26	.244	.042	.0163	.012	.0163	.042	.244	1.26
	200	25	1.27	.261	.084	.261	1.27	.261	.084	.261	1.27
	800	40	.475	.204	.054	.0193	.0085	.0047	.0029	.0023	.0022
	400	40	.476	.206	.057	.024	.017	.024	.057	.206	.476
	200	40	.486	.230	.114	.230	.486	.230	.114	.230	.486
Flame Arc Inclined	800	25	3.81	.265	.034	.010	.004	002	.001	.001	.001
	400	25	3.81	.266	.035	.012	.008	.012	.035	.266	3.81
	200	25	3.82	.276	.07	.276	3.82	.276	.07	.276	3.82
	800	40	1.49	.32	.055	.0165	.0064	.0032	.0021	.0017	.0016
	400	40	1.49	.32	.057	.0168	.0128	.0168	.057	.32	1.49
	200	40	1.50	.34	.112	.34	1.50	.34	.112	.34	1.50
6.6 Amp. A C. Enclosed	800	25	.28	.032	.0044	.0016	.0007	.0003			
	400	25	.28	.032	.0044	.0016	.0014	.0016	.0044	.032	.28
	200	25	.28	.035	.009	.035	.28	.035	.009	.035	.28
	800	40	.11	.042	.008	.0024	.0011	.0005	.0003		
	400	40	.11	.042	.008	.003	.0022	.003	.008	.042	.11
	200	40	.11	.045	.016	.045	.11	.045	.016	.045	.11
350 C. P. Tungsten	800	25	.37	.063	.0095	.003	.0012	.0006	.0004	.0002	.0002
	400	25	.37	.032	.01	.004	.0024	.004	.01	.032	.37
	200	25	.37	.038	.025	.038	.37	.038	.025	.038	.37
200 C. P. Tungsten	400	20	.362	.035	.011	.005	.003	.005	.011	.035	.362
	200	20	.365	.04	.023	.04	.365	.04	.023	.04	.365
	d	h	0	25	50	75	100	125	150	175	200
100 C. P. Tungsten	200	20	.181	.078	.015	.0068	.005	.068	.018	.078	.181
	100	20	.183	.084	.034	.084	.183	.084	.034	.084	.183
60 C. P. Tungsten	200	20	.11	.046	.01	.004	.003	.004	.01	.046	.11
	100	20	.11	.052	.021	.052	.11	.052	.021	.052	.11
32 C. P. Tungsten	200	20	.06	.024	.005	.002	.002	.002	.005	.024	.06
	100	20	.09	.028	.010	.028	.06	.028	.010	.028	.06

VI. STREET LIGHTING

Street lighting in different parts of a city must serve various purposes. For utility and safety, the business streets require a different treatment from the residence streets and suburban districts. Parks and open places require a still different arrangement of lights. A good division of the city is the following:

A Principal business streets;
B Important cross streets and boulevards;
C Residence streets;
D Outlying districts.

14. *Business Streets.*—In the principal business streets, the illumination should be more or less brilliant. It should be as uniform as possible, and of sufficient intensity to enable one to read ordinary sized print, i. e., at least .25 to 1 foot candle. In many cities in Europe, the inclined carbon flame arc is used for these streets. The lamps are suspended over the center of the street from tall ornamental poles on both sides, and at a height of from 40 to 50 ft. They are spaced from four to five times their height. These produce a very beautiful effect, and the street is brightly illuminated. With the reflection from buildings, the illumination averages about two foot candles. This lighting is also carried out for short distances into the cross streets. In this country, the flame arc has not met with favor on account of the high operating cost for the short burning arc. With the advent of the more modern long burning flame arc, this objection can not be made, since, in lighting from large units spaced comparatively near together, the number of the lights will be nearly inversely to the mean lower hemispherical candle power. This will be more apparent later. The standard for such lighting has not yet been set in this country. There is now a strong tendency towards clusters of tungsten lights, or arc lights on ornamental poles, placed low on both sides of the street. It is claimed justly that the best lighted streets attract the crowds and hence merchants are insisting upon brighter and brighter illumination on their streets, even assisting the city in meeting the additional cost of maintenance.

15. *Cross Streets and Boulevards.*—The boulevards and cross streets require a much less brilliant illumination. The principal cross streets do not usually have shade trees, and may be lighted either by high intensity lamps suspended high or low intensity lamps more closely spaced and supported low. The general characteristics of the streets will indicate the type of lighting to be used. In such streets, heavy slow moving traffic is usual. The

boulevards, on the other hand, are usually shaded and require the lights to be suspended low unless the streets are very wide. For shaded streets, lights upon ornamental poles at the sides of the streets are preferable to those suspended over the center, on account of the character of the street. By this is meant that since such a street is used mostly for light vehicles, moving more or less rapidly, lights should not be hung so as to blind the drivers. The appearance of such a street is improved by side lighting, or lights suspended fairly high over the center of wide streets. Probably from .10 to .5 foot candle is a good figure to use.

16. *Residence Streets.*—Residence streets should have one light at each street crossing, for the safety of vehicles and pedestrians approaching these crossings, as collisions are more liable to occur at such points. These streets are usually rather narrow and densely shaded, thus requiring the lights to be supported not over 15 to 25 ft. above the street surface. The maximum illumination should not be too high or the minimum too low. The general average of the illumination should, however, be low. The character and color of the surface has a marked effect upon the lighting of such streets. It is well known that a street with snow upon its surface appears much better lighted than the same one when the snow has disappeared. It is usual in dim illumination to see nearly all obstructions as dark objects against the lighter background of the lighted street. Such a contrast is more distinct if the background is lighter. A macadamized road or asphalt pavement needs much more illumination than one paved with light bricks. It will be seen from Tables 4 to 13 that lights will not give good illumination at the middle of blocks 400 ft. long, if the lights are 25 ft or less above the street. The illumination in the immediate neighborhood of most arc lights is also higher than necessary for this class of street. In crooked streets or where the surface is uneven or hilly, lights must be spaced more closely than at every street crossing. Residence street lighting is the particular field for the incandescent light. In choosing a light for this class of service, the economic limit comes in the ordinary spacing of poles on the street, and the lowest value of illumination allowable immediately below the light. A street having many small lights, hung low and spaced rather closely together, has long been associated with the narrow street of the poorer quarter with its dingy appearance. The brilliant effect and white light from the tungsten light has done much to dispel this prejudice.

17. *Outlying Districts.*—In the outlying districts, either the tungsten light or an arc light having low operating cost may be used. It is seldom that a community can afford to have an arc light at each street intersection in such a district. Hence for protection and safety, the tungsten light may be used to good advantage. A difference here may also be made between streets connecting trade centers and cross streets. The former may be treated as boulevards or as residence streets, according to the traffic conditions.

It is thus seen that no specific rules can be laid down in street lighting. Each case to be handled is more or less special. However in choosing a general lighting scheme for any city, the central station furnishing the power must also be considered. It would of course, not be feasible to specify a variety of lights, requiring varied generating apparatus. In any system, the station apparatus should be interchangeable from one circuit to another for continuity of service and low first cost and maintenance. Thus, if D. C. lamps are chosen, they should all have the same current consumption and this should also fit well with incandescent lamps on the market. On the other hand small compensators may be supplied with each A. C. lamp to adapt it to any A. C. circuit. Wires in series circuits have a voltage upon them at the station, dangerous to personal safety. The circuit may become accidentally grounded at some point, either by contact with a telephone wire or the limb of a tree when wet. In such case, the full voltage may exist between the other wire and ground. Hence, for safety to persons, these wires should not be allowed too near the ground. They are usually required to be strung on the upper cross arm of the poles. If they are brought down to incandescent lamps, suspended too low or strung on low poles, serious accidents are liable to occur. To avoid this, especially low poles should not be used for series lighting. For such service, tungsten multiple 110 volt lamps may be used from special feeder circuits or series lights on ornamental posts fed from armored cables buried in the ground. This latter applies especially to ornamental park lighting.

VII. COST OF OPERATION

The choice of a lamp for a given class of street lighting depends upon the illumination which may be obtained from the lamp and upon its cost of operation. A flat rate is usually made for street lighting; i. e., a charge of so many dollars per lamp per year. The items which enter into a consideration of these charges are:

A Fixed charges;
B Maintenance charges;
C Energy charge.

These items, together with the number of hours in operation, or schedule, determine the rate to be charged for the service. These will be considered in the order named.

18. *Fixed Charges.*—In this item is considered a charge of 6 per cent, covering interest on the capital invested, taxes, insurance, etc., and a depreciation charge of 10 per cent. This latter charge, if deposited at 5 per cent in a sinking fund, would replace the equipment in from eight to nine years. With the present rapid advance in the development of lamps, this figure seems conservative. It means that replacement is due to advance in the art, rather than to the complete deterioration of the apparatus. On the other hand, the depreciation charge depends also upon the life of the city ordinance. If the ordinance is for five years, the depreciation should be 18 per cent instead of 10 per cent, since the city may require an entire change of its system at the end of the contract or award the contract to some other competing company.

19. *Maintenance Charges.*—In the following table, this charge is itemized and computed on the basis of 1000 hours' operation. It covers the cost of renewals, due to the consumption of electrodes, breakage of glassware, repairs to the mechanism of the lamp, and a charge for labor in trimming, cleaning of glassware and reflectors, inspection, store room charges, etc. These figures represent an average installation, say of 400 arc lamps or their equivalent in incandescent lamps. The trimming cost for tungsten lamps is low on account of the long life of these units. The arc lamps are supposed to be cleaned and inspected by the trimmer. In the case of the tungsten lamps, it is assumed that the reflectors will be thoroughly inspected and cleaned each time the lamps are renewed and also at regular intervals, say four times per year, independent of the schedule. Hence this cleaning charge is placed as an annual charge instead of on the basis of 1000 hours.

20. *Energy Charge.*—Energy is usually computed upon the basis of the kilowatt hours consumed. For convenience of comparison, the cost of energy at the lamp terminals, rather than at the station switchboard, is considered here. The energy charge will then cover the cost of delivering this power, such as depreci-

ation on station apparatus, poles, wires, etc., and the maintenance charges on the same. In order to compare direct and alternating current lamps operated from the same station, the cost of the rectifier outfit is included in the cost of the direct current lamps, and its depreciation charge is placed there also. The efficiency of a rectifier outfit may be taken as 87.5 per cent. The loss of energy must therefore be charged to the D. C. lamps. This charge is made by increasing the cost of energy in line 31, Table 15, inversely in the ratio of the efficiency to 100 per cent. These values are then used in Tables 17 and 18 and in line 31, Table 15. No accurate information is available as to the energy charge for the different stations of this State. Tables 15 and 16 have been computed with an energy charge of one cent per kilowatt hour. From Tables 15 and 16, Tables 17 and 18 have been computed for two schedules and an energy charge from one cent to five cents per kilowatt hour. Curves have also been drawn between energy charge and cost of operation per lamp per year for two different schedules. These are shown in Fig. 25 to 28. They show a ready comparison between the operating cost of the different lamps chosen. There are many factors which affect the cost of energy in different cities. Some of these are the following:

A Cost of coal delivered at the power plant;

B Available water power or natural gas;

C Cost of the land on which the plant is located and taxes on the same;

D Amount of power developed and the load factor;

E Available market for steam heat;

F Cost of delivering the power to the consumer, (under this item will come type of construction, whether pole lines or conduits and taxes or rentals on the same);

G Interest and depreciation on the equipment.

These items vary so widely in different localities that the writers have not attempted to make any definite statements.

21. *Schedules.*—There are numerous schedules in use in this country. The "all night and every night", or 4000 hours per year schedule is used in most cities having a population of 20,000 and over. In smaller cities and suburban districts, various moonlight schedules are in use, the average of these being about 2500 hours per year. This latter schedule is supposed to approximate moonlight every night in the year. The intensity of illumination at the first or last quarter of the moon is only about one-tenth of that

at full moon instead of one-half as might be supposed. Hence the moonlight schedule should be more nearly three-fourths of the all night schedule, instead of from one-half to five-eighths that amount. In small cities, where only a few circuits are necessary, and where a duplicate set of station equipment cannot be afforded, the all night and every night schedule may be approximated by allowing four nights each month near full moon when the lights on certain circuits may be turned off, allowing the necessary repairs to be made. Many other schedules are also in use but are not listed here. Calculations covering the cost of operation of lamps may be made in a manner similar to those for Tables 17 and 18.

Cost Tables

To illustrate how the values in Table 17 are derived the following problem is chosen.

Problem.—Assume a 6.6 amperes, alternating current, enclosed arc lamp to be operating on a 4000 hour schedule, the energy costs being three cents per kilowatt hour. What will be the cost of operating this lamp for one year?

From Table 4.—

For 1000 Hours and One Cent per Kilowatt Hour, (See Table 15):

Annual fixed charges	$3.20
Maintenance per 1000 hours	2.30
Energy per 1000 hours at one cent	4.25

For 4000 Hours at Three Cents per Kilowatt Hour,

Annual fixed charges	$3.20
Maintenance charges for 4000 hours $=4\times2.30=$	9.20
Energy for 4000 hours at three cents $=4\times3\times4.25=$	51.00
Total charge per lamp per year,	$63.40

A total charge of $70.00 per lamp per year would represent a fair profit to the central station.

An analysis of the three factors entering into the operating cost of street lamps may be made from Tables 15 to 18. It is at once apparent that none of the factors may be omitted safely for any of the lamps. The greater the number of hours burning per year, the less the importance of the fixed charge. On the other hand, the shorter the schedule, the more important the fixed charge, and the less the importance of the maintenance and energy charges. Another fact clearly shown is the effect upon annual cost of the length of life of the electrodes per trim. The

TABLE 15

COST AND OPERATING CHARACTERISTICS OF ARC LAMPS

	ITEMS	Open Arc D. C. Twin Carbon	Enclosed Arc. D. C.	Enclosed Arc A. C.	Enclosed Arc A. C.	Magnetite Arc D. C.	Magnetite Arc D. C.	Flame Arc D. C. Inclined Carbon	Flame Arc A. C. Inclined Carbon	Flame Arc D. C. Vertical Carbon Long Burning	Flame Arc A. C. Vertical Carbon Long Burning
1	Voltage across lamp terminals	50	72	72	72	80	80	55	55	70	70
2	Current	9.6	6.6	7.5	6.6	6.6	4.0	10	10	5.5	5 5
3	Terminal watts	480	475	480	425	528	320	550	167	385	350
4	Maximum candle power value	1250	525	350	275	1600	730	2400	—	2575	1925
5	Maximum candle power angle with vertical	45°	65°	40°	40°	80°	80°	0°-30°	—	70°	85°
6	Mean lower hemispherical candle power (H. S. C. P.)	800	468	298	244	1170	457	1785	850	1800	1485
7	Watts per H. S. C. P.	.6	1.01	1.61	1.71	.45	.70	.31	.55	.26	.24
8	Kind of glassware, inner globe	None	Opal	Opal	Opal	None	None	None	None	Clear	Clear
9	" " " outer "	Clear	Clear	Clear	Clear	Clear	Clear	Opal	Clear	Opal	Clear
10	Curve sheet, figure	3 A	4 C	4 B	4 A	3 B	3 C	5 D	—	5 C	—
11	Life positive electrode, hours	17	125	100	110	1500	4000	17	17	70	100
12	" negative " "	17	125			100	175	17		70	100
13	" inner globe	—	400	400	400	—	...	—	—	300	300
14	" outer "	1000	2000	2000	2000	2000	2000	1000	1000	2000	2000
15	Cost of lamp, dollars	15.00	34.50	20.00	20.00	40.00	40.00	60.00	60.00	60.00	60.00
16	" " positive electrodes, dollars per trim	.015	.021	.022	.022	.50	.45	.14	.14	.25	.25
17	Cost of negative electrodes, dollars per trim					.06	.05				
18	Cost of inner globe, dollars each	—	.15	.15	.15	—	—	—	—	.50	.50
19	" " outer " " "	.50	.50	.50	.50	1.00	1.00	1.00	1.00	.75	.75
20	" " labor in trimming, etc., per trim	.06	.06	.06	.06	.06	.06	.06	.06	.06	.06

	A. FIXED CHARGES, ANNUAL										
21	Interest on investment at 6%	.90	2.07	1 20	1.20	2.40	2.40	3.60	3.60	3.60	3.60
22	Depreciation at 10%	1.50	3.45	2.00	2.00	4.00	4.00	6.00	6.00	6.00	6.00
23	Total annual fixed charge	2.40	5.52	3.20	3.20	6.40	6.40	9.60	9.60	9.60	9.60
	B. MAINTENANCE CHARGES PER 1000 HOURS										
24	Cost of electrodes, dollars	.80	.17	.22	.20	.93	.40	8.23	8.23	3.56	2.50
25	" " inner globes, dollars	—	.37	.37	.37	—	—	—	—	1.67	1.67
26	" " outer " "	.50	.25	.25	.25	.50	.50	1.00	1.00	.37	.37
27	" " trimming and inspection, dollars	3.54	.48	.60	.55	.60	.31	3.54	3.54	.85	.60
28	Cost of rectifier renewals, dollars	—	.50	—	—	.50	.50	.50	—	.50	—
29	" " repairs, dollars	.50	.50	.50	.50	.50	.50	.50	.50	.50	.50
30	" " maintenance, total, dollars	5.43	2.27	1 94	1.87	3.03	2.24	13.77	13.27	7.45	5.64
	C. COST OF ENERGY, PER 1000 HOURS										
31	At one cent per kilowatt hour	4 80	5.42	4.80	4.25	6 05	3.65	6.25	4.67	4.40	3.50

* Line 31 has been computed from line 3, considering the rectifier efficiency.

TABLE 16

COST AND OPERATING CHARACTERISTICS OF TUNGSTEN LAMPS

	ITEMS	For Suspension in Place of Arc					For Suspension at Side of Street				
1	Candle power horizontal	32	60	100	200	350	32	60	100	200	350
2	Terminal watts	38	71	118	236	413	38	71	118	236	413
3	Watts per mean horizontal candle power	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18
4	Mean hemispherical candle power (with reflector)	43	80	135	227	471	43	80	135	227	471
5	Watts per H. S. C. P.	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
6	Maximum candle power, value	45	84	140	280	490	45	84	140	280	490
	Maximum candle power, angle with vertical	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
7	Curve sheet, figure										
8	Life of lamp, hours	1350	1350	1350	1350	1350	1350	1350	1350	1350	1350
9	Cost of lamp, dollars	1.00	1.00	1.20	2.44	3.75	1.00	1.00	1.20	2.44	3.75
10	" " labor for inspection and cleaning, annual	.40	.40	.40	.40	.40	.40	.40	.40	.40	.40
11	Cost of fixtures, dollars	1.50	1.50	2.00	2.50	2.50	5.00	5.00	5.00	5.00	5.00
12	" " lamp and fixtures	2.50	2.50	3.20	4.94	6.25	6.00	6.00	6.20	7.44	8.95
	A. FIXED CHARGES, ANNUAL										
13	Interest on investment at 6%	.15	.15	.17	.30	.37	.36	.36	.37	.45	.52
14	Depreciation at 10%	.25	.25	.32	.49	.63	.60	.60	.62	.74	.87
15	Annual inspection charge	.40	.40	.40	.40	.40	.40	.40	.40	.40	.40
16	Total annual fixed charges	.80	.80	.89	1.19	1.40	1.36	1.36	1.39	1.59	1.79
	B. MAINTENANCE CHARGES PER 1000 HOURS										
17	Cost of lamp renewals, dollars	.74	.74	.83	1.80	2.80	.74	.74	.83	1.80	2.80
18	" " repairs	.05	.05	.05	.10	.10	.15	.15	.15	.15	.15
19	" " maintenance total	.79	.79	.88	1.90	2.90	.89	.89	.98	1.95	2.95
	C. COST OF ENERGY PER 1000 HOURS										
20	At one cent per kilowatt hour	.38	.71	1.18	2.36	4.13	.38	.71	1.18	2.36	4.13

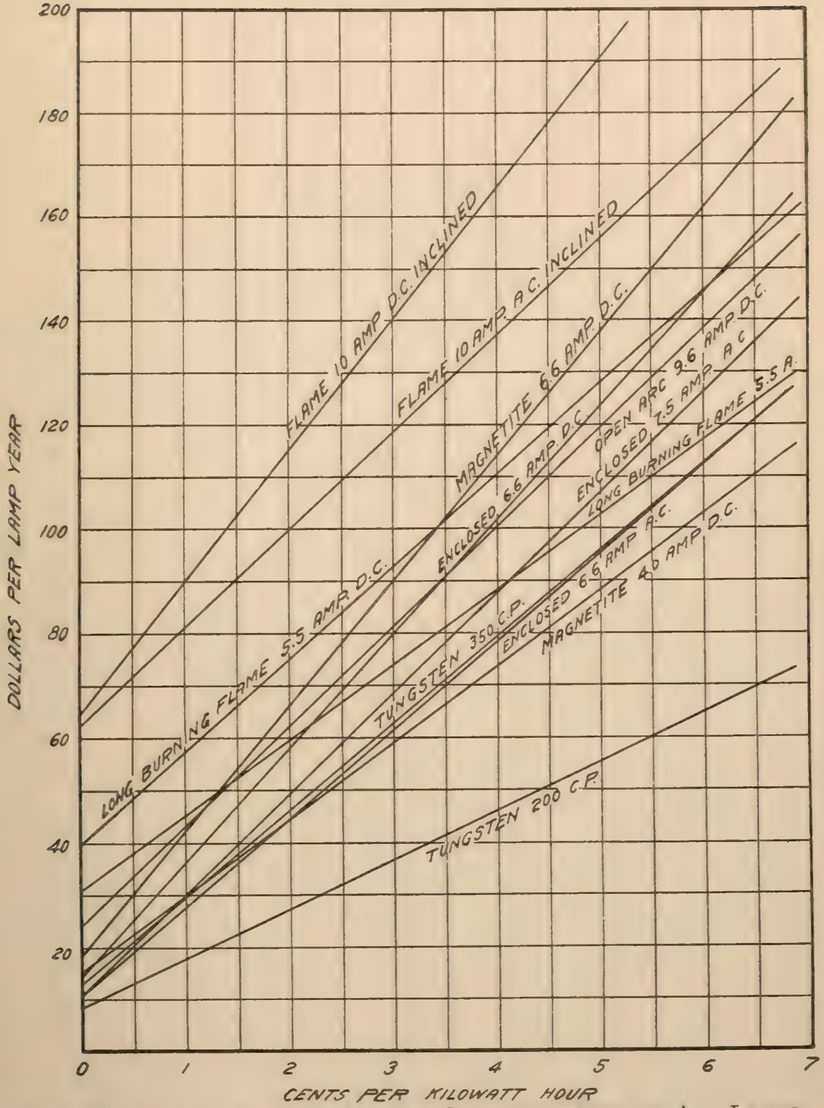

FIG. 25. CURVES SHOWING ANNUAL OPERATING COST OF ARC LAMPS ON 4000–HR. SCHEDULE

TABLE 17

OPERATING COST ON 4000-HOUR SCHEDULE

	ITEMS	Open Arc Twin Carbon D. C.	Enclosed D. C.	Enclosed A. C. 7.5 Amp.	Enclosed A. C. 6.6 Amp.	Magnetite 6.6 Amp.
	FOR 1000 HOURS FROM TABLE					
1	Annual fixed charge......	2.40	5.52	3.20	3.20	6.40
2	Maintenance for 1000 hours......	5.42	2.27	1.94	1.87	3 03
3	Energy per 1000 hours at 1 cent per K. W. Hr...	4.80	5.42	4.80	4.25	6.05
	ANNUAL COST 4000 HOUR SCHEDULE					
4	Annual fixed charge (1)...........................	2.40	5.52	3.20	3.20	6.40
5	Maintenance, 4 × (2)	21.68	9.08	7.76	7.48	12.12
6	Energy at 1 cent, 4 × (3).........................	19.20	21.70	19 20	17.00	24.20
7	" " 2 cents, 2 × 4 × (3).......	38.40	43.40	38.40	34.00	48.40
8	" " 3 " , 3 × 4 × (3).....................	57.60	65.20	57.60	51,00	72.60
9	" " 4 " , 4 × 4 × (3).....................	76 80	86.80	76.80	68.00	96.80
10	" " 5 " , 5 × 4 × (3).....................	96.00	108.50	36.00	85.00	111.00
	TOTAL ANNUAL OPERATING COST					
11	Total cost per lamp, 1 cent per K. W. Hr. (4) + (5) + (6)	43.28	36.30	30.16	27.68	42.72
12	Total cost per lamp, 2 cents per K. W. Hr. (4) + (5) + (7)	62.48	58.00	49.36	44.68	66.92
13	Total cost per lamp, 3 cents per K. W. Hr. (4) + (5) + (8)	81.68	79.80	68.56	61.68	91.12
14	Total cost per lamp, 4 cents per K. W. Hr. (4) + (5) + (9)	100.88	101 40	87.76	78.68	105.32
15	Total cost per lamp, 5 cents per K. W. Hr. (4) + (5) + (10)	120.08	123.10	106.96	95.68	129.52

TABLE 17 (*Continued*)

4000-HOUR SCHEDULE

					Center Suspension					Side Suspension				
Magnetite 4.0 Amp.	Flame D. C. Inclined	Flame A. C. Inclined	Flame D. C. Vertical Long Burning	Flame A. C. Vertical Long Burning	Tungsten 32. C. P.	Tungsten 60 C. P.	Tungsten 100 C. P.	Tungsten 200 C. P.	Tungsten 350 C. P.	Tungsten 32 C. P.	Tungsten 60 C. P.	Tungsuen 100 C. P.	Tungsten 200 C. P.	Tungsten 350 C. P.
6.40	9.60	9.60	9.60	9.60	.80	.80	.84	1.19	1.40	1.36	1.36	1.39	1.59	1.79
2.24	13.77	13.27	7.45	5 64	.79	.79	.88	1.90	2.90	.89	.89	.98	1.95	2.95
3.65	6.25	4.67	4.40	3.50	.38	.71	1.18	2.36	4.13	.38	.71	1.18	2.36	4.13
6.40	9.60	9.60	9.60	9.60	.80	.80	.84	1.19	1.40	1.36	1.36	1.39	1.59	1.79
8.96	55.08	53.08	29.80	22.56	3.16	3.16	3.52	7.60	11,60	3.56	3.56	3.92	7.80	11.80
14.60	25.20	18.68	17.60	14.00	1.52	2.84	4.72	9.44	16.52	1.52	2.84	4.72	9.44	16.52
29.20	50.40	37.36	35.20	28.00	3.04	5.68	9.44	18.88	33.04	3.04	5.68	9.44	18.88	33.04
43.80	75.60	56.04	52.80	42.00	4.52	8.52	14.16	28.32	49.56	4.56	8.52	14.16	28.32	49.56
58.40	100.80	74.72	70.40	56.00	6.08	11.36	18.88	37.76	66.08	6.08	11.36	18 88	37.60	66.08
73.00	126.00	93.40	86.00	70.00	7.60	14,20	23.60	47.20	82.60	7.60	14.20	23.60	47.20	82 60
29.96	89.88	81.36	57.00	46.16	5.48	6.80	9.08	18.23	29.52	6.44	7.76	10.03	18.83	30.11
34.56	115.08	100.04	74.60	60.16	7.00	9.64	13.80	27.67	46.04	7.96	10.60	14.75	28.27	46.63
59.16	140.28	118.72	92.20	74.16	8.52	12.48	18.52	37.11	62.56	9.48	13.44	19.47	37.79	63.15
73.76	165.48	137.40	109.80	88.16	10.04	15.32	23.24	46.55	79.08	11.00	16.28	24.19	47.15	79.67
88.36	190.68	156.08	125.40	102.16	11.56	18.16	27.96	55.99	95.60	12.52	19.12	28.91	56.59	96.19

TABLE 18

OPERATING COST ON 2500-HOUR SCHEDULE

	Items	Open Arc Twin Carbon D. C.	Enclosed D. C.	Enclosed A. C. 7.5 Amp.	Enclosed A. C. 6.6 Amp.	Magnetite 6.6 Amp.
	FOR 1000 HOURS FROM TABLE					
1	Annual Fixed Charge....	2.40	5.52	3.20	3.20	6.40
2	Maintenance For 1000 hours....	5.42	2.27	1.94	1.87	3.03
3	Energy Per 1000 hours at 1 cent Per K. W. Hour	4.80	5.42	4.80	4.25	6.05
	ANNUAL COST 2500 HOUR SCHEDULE					
4	Annual Fixed Charge (1)	2.40	5.52	3.20	3.20	6.40
5	Maintenance, 2.5 x (2)....	13.55	5.67	4.85	4.67	7.57
6	Energy at 1 cent, 2.5 x (3)....	12.00	13.55	12.00	10.62	15.10
7	Energy at 2 cents, 2.5 x 2 x (3)....	24.00	27.11	24.00	21.24	30.20
8	Energy at 3 cents, 2.5 x 3 x (3)....	36 00	40.70	36.00	31.86	45.30
9	Energy at 4 cents, 2.5 x 4 x (3)	48.00	54.30	48.00	42.48	60.40
10	Energy at 5 cents, 2.5 x 5 x (3)....	60.00	67.70	60.00	53.10	75.50
	TOTAL ANNUAL OPERATING COST					
11	Total Cost Per Lamp, 1 cent Per K. W. Hr. (4) + (5) + (6)	27.95	24.74	20.05	18.49	29.07
12	Total Cost Per Lamp, 2 cents Per K. W. Hr. (4) + (5) + (7)	39.95	38.30	32.05	29.11	44.17
13	Total Cost Per Lamp, 3 cents Per K. W. Hr. (4) + (5) + (8)....	51.95	51.89	44.05	39.73	59.27
14	Total Cost Per Lamp, 4 cents Per K. W. Hr. (4) + (5) + (9)....	63.95	65.49	56.05	50.35	74.37
15	Total Cost Per Lamp, 5 cents Per K. W. Hr. (4) + (5) + (10)....	75.95	78.89	68.05	60.97	89.47

TABLE 18 (*Continued*)

2500-Hour Schedule

Magnetite 4.0 Amp.	Flame D. C. Inclined	Flame A. C. Inclined	Flame D. C. Vertical Long Burning	Flame A. C. Vertical Long Burning	Center Suspensions					Side Suspension				
					Tungsten 32 C. P.	Tungsten 60 C. P.	Tungsten 100 C. P.	Tungsten 200 C. P	Tungsten 350 C. P.	Tungsten 32 C. P.	Tungsten 60 C. P.	Tungsten 100 C. P	Tungsten 200 C. P.	Tungsten 350 C. H,
6.40	9.60	9 60	9.60	9.60	.80	.80	.84	1.19	1.40	1.36	1.36	1.39	1.59	1.79
2.24	13.77	13.27	7.45	5.64	.79	.79	.88	1.90	2.90	.89	.89	.98	1.95	2.95
3.65	6.25	4.67	4.40	3.50	.38	.71	1.18	2.36	4.13	.38	.71	1.18	2.36	4.13
6.40	9.60	9.60	9.60	9.60	.80	.80	.84	1.19	1.40	1.36	1.36	1.39	1.59	1.79
5.60	34.42	33.11	18.62	14.10	1.97	1.97	2.20	4.75	7.25	2 22	2.22	2.45	4.87	7.37
9.14	15.70	11.67	11.00	8.75	.95	1.77	2.95	5.90	10.32	.95	1.77	2.95	5.90	10.32
18.30	31.40	23.34	22.00	17.50	1.90	3.54	5.90	11.80	20.64	1.90	3.54	5.90	11.80	20 64
27.40	47.10	35.01	33 00	26.25	2.85	5.31	8 85	17.70	30 96	2.85	5.31	8.85	17.70	30.96
36.50	62.80	46.68	44.00	35.00	3.80	7.08	11.80	23.60	41.28	3.80	7.08	11.90	23.60	41.28
45.70	78.60	58.35	55.00	43.75	4.75	8.85	14.75	29 50	51.60	4.75	8.85	14.75	29.50	51.60
21.14	59.72	54.44	39.22	32.45	3.72	4.54	5.99	11.84	18.97	4.53	5.35	6.79	12.36	19.48
30.30	75.42	66.11	50.02	41.20	4.67	6.31	8.94	17.74	29.29	5.48	7.12	9.74	18.26	29.80
39.40	91.12	77.78	61.22	49.95	5.62	8.08	11.89	23.64	39.61	6.43	8.89	12.69	24.16	40.12
48.50	106.82	89.45	72.22	58.70	6.57	9.85	14.84	29.54	49.93	7.38	10.66	15.64	30.06	50.44
57.70	122.62	101.12	83.22	67.45	7.52	11.62	17.79	35.44	60.25	8.33	12.43	18 59	35.96	60.76

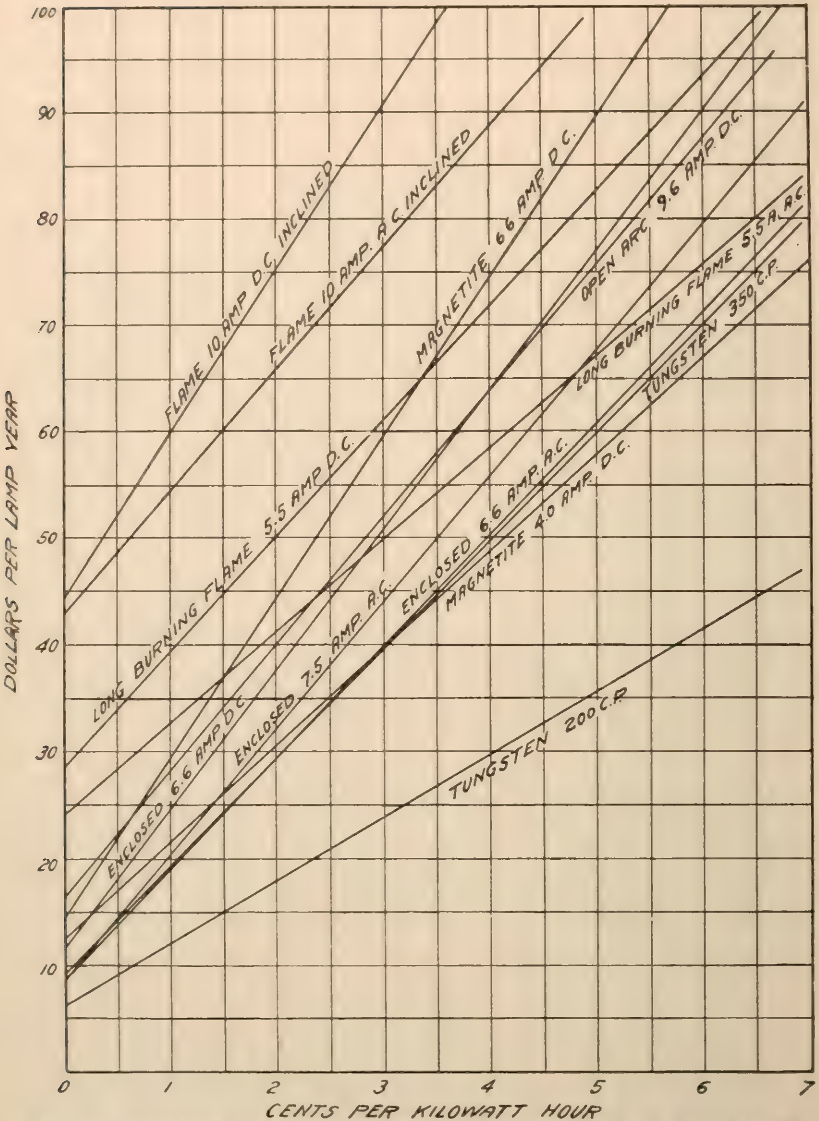

FIG. 26. CURVES SHOWING ANNUAL OPERATING COST OF ARC LAMPS ON 2500-HR. SCHEDULE

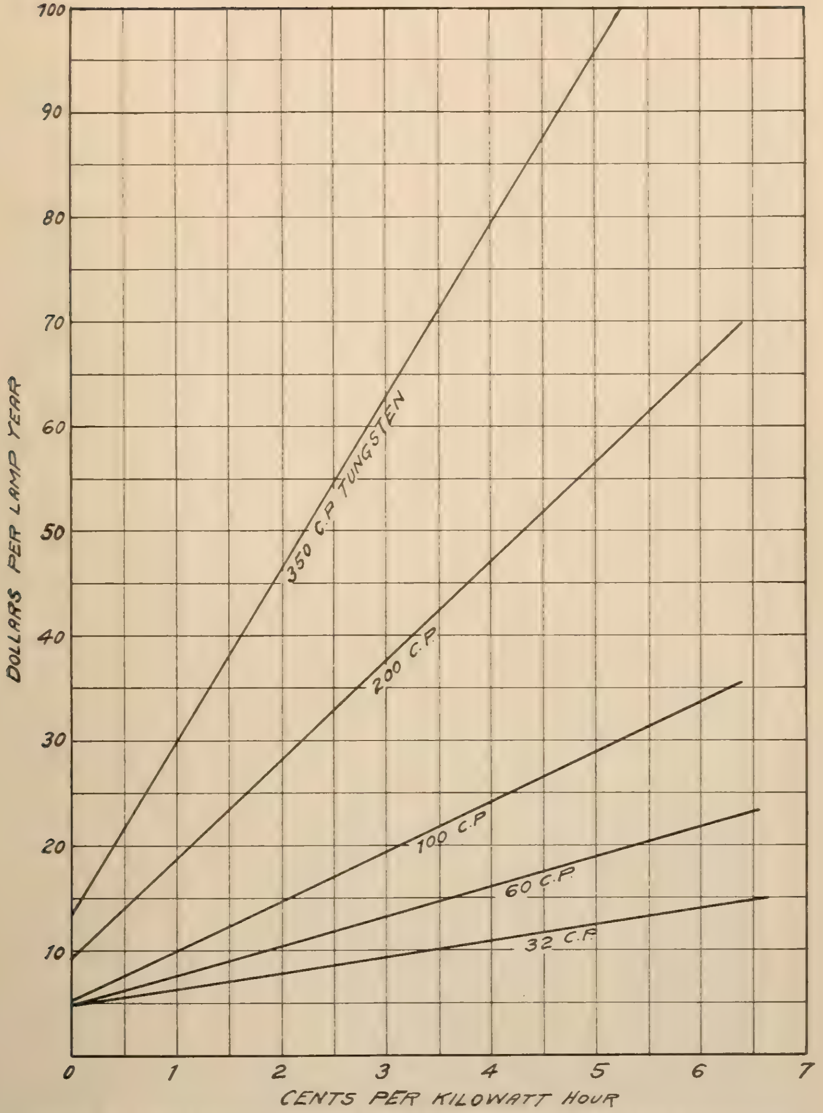

FIG. 27. CURVES SHOWING ANNUAL OPERATING COST OF TUNGSTEN LAMPS ON 4000-HR. SCHEDULE

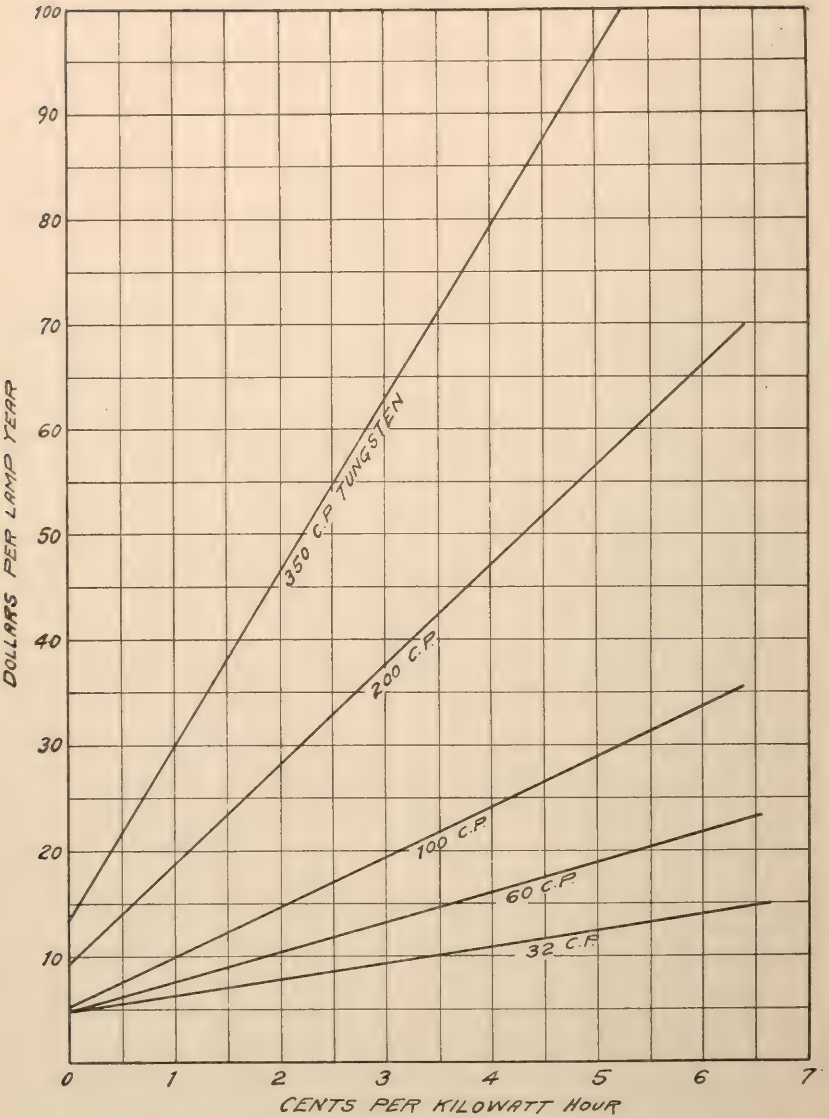

FIG. 28. CURVES SHOWING ANNUAL OPERATING COST OF TUNGSTEN LAMPS ON 2500-HR. SCHEDULE

item of labor and supplies is large for the open arc, and for short burning flame arcs. In fact, the maintenance charge on these lamps is nearly equal to the total operating charge on some of the other lamps. This is not offset by a proportionately greater amount of light. In the tungsten lamps, the energy charge is rather more important than in most of the arc lamps, but the fixed and maintenance charges are low on account of the long life of these lamps. Although the cost of each renewal is rather high, only very few renewals have to be made each year.

In the case of cities where a lighting system using series lights is already installed, the question arises as to the proper rate to be agreed upon in a new ordinance. This is usually computed on the basis of the lights at present in use, or these lights replaced by the same number of lamps of some different type. In fixing the rate for these new lights, Tables 17 and 18 or Fig. 22 to 28 will be of direct application, since the rate will be in proportion to the cost per lamp per year. A comparison of the lamps on the basis of street distribution will show whether the new lamp is equivalent to the old one in illuminating power. It will be seen that with four cent energy, a saving of $4 per lamp per year may be expected by replacing 6.6 amperes A. C. lamps with 4.0 ampere magnetites. At the same time, the street illumination is improved. For 100 lamps, this means a saving of $400 per year on the 4000 hour schedule. The 200 C. P. tungsten light would give as good satisfaction as the 6.6 amperes A. C., on account of its steady light, and at a saving of $3200 per year for the 100 lights.

It has been shown, from the comparison of Tables 17 and 18, that the high cost of maintenance for short burning lamps makes their yearly cost very high on any schedule or cost of energy, in comparison with the long burning lamps. This can be compensated for only by the use of fewer lamps. It will be seen, however, that for residence districts, the illumination is so low at the minimum point that none of the lamps can be spaced at a greater distance than 400 ft., or one block. Hence the comparison of arc lamps for this class of service must be made upon this spacing. The illumination on such streets can be improved only by using smaller units spaced closer together. This is the special field for the tungsten lamps. To show this more clearly, the following comparison has been made.

Problem.—To light a shaded residence street one mile long, 400 ft. blocks, energy being assumed at three cents per kilowatt hour and the 4000 hour schedule. Using only the lower cost lamps, Table 19, page 59, has been computed. It is seen that a good minimum and low maximum illumination can be produced by using tungsten lamps. There will be required 14 lamps at 400 ft. or 27 lamps at 200 ft. or 53 lamps at 100 ft. This is assuming that poles are spaced conveniently for this arrangement. It will be seen that nearly as good illumination is produced by 60 C. P. tungstens, spaced 100 ft. apart, as by any of the arc lights, except the 6.6 amperes magnetite, and at a cost of only $713 per year, compared with $3220 for the 6.6 A. C. enclosed. Probably the 200 C. P. tungsten would be selected at 200 ft. spacing, costing $1020, since the street would appear much more brilliantly lighted than with the 60 C. P. Other comparisons will doubtless suggest themselves, depending upon local conditions.

VII. Summary and Conclusions

From Tables 1, 2 and 3, it is seen that for uniform distribution of illumination on a surface from single light sources, their illumination not overlapping to any extent requires very low values of intensity at angles near the vertical, and to as wide an angle as 60°, except where the lights are suspended high over the surface to be illuminated. A comparison of the curves in Fig. 3, 4 and 5 for arc lamps with these tables shows that none of these distribution curves can be expected to give uniform low illumination. Curves of the shape of those shown by B or C, Fig. 3, or A or C, Fig. 5, will give wider uniform distribution than the other curves shown. Curve A, Fig. 3, shows that a bright ring of light must be expected at a distance from the base of the lamp equal to the light with a dark circular area within, and a rather abrupt dark area without. For curves in Fig. 4, the lamps must be spaced a little farther apart than their height. A considerable amount of light between 60° and 90° is available for overlapping. Curves B and D, Fig. 5, and the curves on Fig. 6 show that these lamps must be used spaced rather close together. Inclined carbon flame lamps can be used successfully only for very bright illumination, on account of the intensity at the zero angle. For dim uniform illumination, the tungsten light is the proper one to chose, on account of the lower intensity available in the small units and its excellent distribution curve for this purpose. This point is very strikingly brought out by comparing the illumination curves in Fig. 22.

A lamp throwing a strong light at a wide angle must be surrounded by an opal globe in order to lower its intrinsic brilliancy. The lower the light is hung over the center of the street, the

TABLE 19

ANNUAL COST OF LIGHTING ONE MILE OF STREET FOR .02-ft. CANDLE HORIZONTAL ILLUMINATION

ITEMS	Illumination in Foot Candles						Annual Cost								For .02 ft. Candle Minimum					
	Spacing 400'		Spacing 200'		Spacing 100'		1 Lamp	Per Mile of Street 4000 Hr. Schedule			1 Lamp	Per Mile of Street 2500 Hr. Schedule			Distance Apart	Lights per Pole	Foot Candle		Cost	
	Max.	Min.	Max.	Min.	Max.	Min.		14 Lamps 400'	27 Lamps 200'	53 Lamps 100'		14 Lamps 400'	27 Lamps 200'	53 Lamps 100'			Max.	Min.	4000 Hour	2500 Hour
Enclosed D. C. 6.6 Ampere	.36	.003	.372	.023	—	—	79.80	1120	2150	4230	51.89	727	1400	2750	200	1	.372	.023	2150	1400
Enclosed A. C. 7.5 Ampere	.336	.0016	.34	.012	—	—	68.56	960	1820	3635	44.05	617	1190	2330	200	2	.68	.024	3640	2380
Enclosed A. C. 6.6 Ampere	.28	.0014	.28	.009	—	—	61.68	865	1660	3270	39.73	556	1083	2100	200	2	.56	.02	3300	2166
Magnetite D. C. 6.6 Ampere	.80	.009	.85	.069	—	—	91.12	1275	2460	4830	59.27	830	1600	3140	400	2	1.8	.018	2550	1660
Magnetite D. C. 4.0 Ampere	.56	.004	.564	.033	—	—	59.16	827	1595	3130	39.40	552	1065	2090	200	1	.564	.033	1595	1065
Flame D. C. 10.0 Ampere Inclined	3.81	.008	3.82	.07	—	—	140.28	2000	3850	7450	91.12	1275	2460	4830	400	2	7.62	.016	1000	2550
Flame D. C. 5.5 Ampere Long Burning	1.26	.012	1.27	.084	—	—	92.20	1290	2490	4880	61.22	857	1650	3240	400	2	2.52	.024	2580	1714
Tungsten 200 C. P. Center Suspension	.362	.003	.365	.023	.366	.068	37.11	520	1000	1968	23.64	331	638	1250	200	1	.365	.023	1000	638
Tungsten 350 C. P. Center Suspension	.37	.002	.37	.025	—	—	62.56	875	1690	3311	39.61	555	1070	2100	200	1	.37	.025	1690	1070
Tungsten 32 C. P. Side Suspension	—	—	.06	.002	.06	.01	9.48	133	256	503	6.43	90	174	341	100	2	.12	.02	1006	682
Tungsten 60 C. P. Side Suspension	—	—	.11	.003	.11	.021	13.44	188	363	713	8.89	126	243	477	100	1	.11	.021	713	477
Tungsten 100 C. P. Side Suspension	—	—	.181	.005	.183	.034	19.47	273	525	1032	12.69	178	342	673	100	1	.183	.034	1032	673
Tungsten 200 C. P. Side Suspension	.362	.003	.365	.023	.366	.068	37.71	528	1020	2000	24.16	338	652	1280	200	1	.365	.023	1020	652
Tungsten 350 C. P. Side Suspension	.575	.002	.6	.02	—	—	63.15	883	1700	3350	40.12	561	1080	2120	200	1	.6	.02	1700	1080

Height 25 ft. for Arcs and 350 C. P. Tungstens.
Height 20 ft. for all others.

more is this important. For very bright lights like the flame arcs, this must also be observed, even when the lights are hung high as they are still in the line of vision and effective at a greater distance. The sentiment of the general public is beginning to demand this consideration more and more each year.

VIII. APPENDIX

DEFINITIONS

Candle-Lumen.—The total flux of light from a source is equal to its mean spherical intensity multiplied by 4π (12.57). The unit of flux is called *lumen.* A *lumen* is the $\frac{1}{4\pi}$ th (0.0796) part of the total flux of light emitted by a source having a mean spherical itensity of one candle-power. A *hefner-lumen* is a 0.90 lumen.

Candle-power.—The luminous intensity of sources of light is expressed in *candle-power.* The unit of candle-power should be derived from the standards maintained by the National Bureau of Standards at Washington, D. C. The *hefner* is 0.90 of this unit. In practical measurements, seasoned and carefully standardized incandescent lamps are more reliable and accurate than the primary standard.

Efficiency of Electric Lamps.—The efficiency of electric lamps is properly stated in mean spherical candle-power per watt, and preferably in lumens per watt at the lamp terminals.

Electrode.—Either of the terminals of an arc lamp between which the arc is formed. The electrodes are sometimes called the carbons.

Foot-candle.—Illumination is expressed in foot-candles. A foot-candle is the normal illumination produced by one unit of candle-power at a distance of one foot.

Mean Horizontal Candle-power.—The average intensity of light in a horizontal plane passing through the center of light source.

Mean Spherical Candle-power.—The average candle-power of a source taken over the surface of a sphere having its center at the source of light. It is numerically equal to the total light emitted divided by 4π (12.57).

Mean Hemispherical Candle power.—The average candle-power in a hemisphere having the center of its plane surface at the center of the light source. Usually the lower hemisphere is chosen.

Note.—The hefner is a unit used mostly in Germany. It is smaller than the candle-power. 1 hefner = .9 candle power.

Moonlight Schedule.—A schedule of burning hours for lamps, the lamps burning only when the moon does not shine; in this bulletin, 2500 hours burning per year.

Watts per Candle-power.—The specific consumption of an electric lamp is its watts consumption per mean spherical candle-power. "Watts per candle" is a term used commercially in connection with incandescent lamps, and denotes watts per mean horizontal candle-power.

Load Factor.—Load factor is the ratio between the average load and the maximum load for the given day.

Reading Distance.—Where standard photometric measurements are impracticable, approximate measurements of illuminants used as street lamps may be made by comparing their "reading distances", i. e., by determining alternately the distances at which an ordinary size of reading print can just be read, by the same person or persons, where all other light is screened. The angle below the horizontal at which the measurement is made should be specified when it exceeds 15°.

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*_Circular No. 1._ High-Speed Tool Steels, by L. P. Breckenridge. 1905. _None available._

*_Bulletin No. 2._ Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks. 1905. _None available._

*_Circular No. 2._ Drainage of Earth Roads, by Ira O. Baker. 1906. _None available._

*_Circular No. 3._ Fuel Tests with Illinois Coal. (Compiled from tests made by the Technologic Branch of the U. S. G. S., at the St. Louis, Mo., Fuel Testing Plant, 1904-1907,) by L. P. Breckenridge and Paul Diserens. 1909. _Thirty cents._

*_Bulletin No. 3._ The Engineering Experiment Station of the University of Illinois, by L. P. Breckenridge. 1906. _None available._

*_Bulletin No. 4._ Tests of Reinforced Concrete Beams, Series of 1905, by Arthur N. Talbot. 1906. _Forty-five cents._

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*_Bulletin No. 9._ An Extension of the Dewey Decimal System of Classification Applied to the Engineering Industries, by L. P. Breckenridge and G. A. Goodenough. 1906. _None available._

*_Bulletin No. 10._ Tests of Concrete and Reinforced Concrete Columns, Series of 1906, by Arthur N. Talbot. 1907. _None available._

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The Board of Trustees of the University of Illinois voted, December 3, 1910, that a price should be affixed to certain University publications, among them being the Bulletin of the Engineering Experiment Station.

This action, so far as it concerns the bulletins of the Engineering Experiment Station, has for its purpose a threefold object:

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(2) To make possible the establishment and maintenance of a trade circulation through the regular publishing houses.

(3) To regulate the distribution of the reserve or "out-of-print" supply.

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(1) All bulletins, hereafter issued, will have a price printed upon the cover, together with the name of the London sales agent.

(2) EACH BULLETIN, HOWEVER, WILL STILL BE SUBJECT TO A FREE INITIAL DISTRIBUTION, AS HERETOFORE, ON THE BASIS OF EXISTING MAILING LISTS.

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(3) THERE WILL BE A LIMITED NUMBER OF COPIES AVAILABLE FOR FREE DISTRIBUTION, UPON REQUEST, AFTER THE INITIAL DISTRIBUTION.

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W. F. M. GOSS
Director.

UNIVERSITY OF ILLINOIS
ENGINEERING EXPERIMENT STATION

BULLETIN NO. 52 DECEMBER 1911

AN INVESTIGATION OF THE STRENGTH OF ROLLED ZINC

BY HERBERT F. MOORE, ASSISTANT PROFESSOR IN THEORETICAL AND APPLIED MECHANICS

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AN INVESTIGATION OF THE STRENGTH OF ROLLED ZINC.

I. Introduction.

1. *Uses of Zinc.*—Zinc is used as a constituent of brass and other alloys, as a protective coating for iron and steel plate and pipe, and as a preventive against rusting out of steam boiler tubes. It is used for making fruit jar covers and corrosion-resisting cans and boxes. In the form of wire, it is used for making shoe nails, and in the form of plates, it is used for making etchings (line cuts) for the reproduction of drawings. Zinc has a wide use as the electro-negative element in electric batteries. The ductility of zinc is an important factor in the manufacture of fruit jar covers, cans, battery zincs, or in other cases when it is to be bent or stamped into shape. Zinc is rarely used as a stress-carrying member of a machine or structure; in Europe thin zinc plates are sometimes used for roofing, and in a few cases electric cables have been suspended by strips of sheet zinc, which resist atmospheric corrosion better than do strips of steel plate.

2. *Purpose of Investigation and Acknowledgment.*—The occasional use of sheet zinc hangers for supporting electric cables called the attention of the Matthiessen and Hegeler Zinc Company of La Salle, Illinois, to the question of the strength of rolled sheet zinc, and it was found that few data were available. The matter was referred to the attention of the Engineering Experiment Station of the University of Illinois, and the general subject of the strength of rolled zinc seemed to be of sufficient importance to warrant making tests. A series of tests of strength of zinc was made in the Laboratory of Applied Mechanics of the University of Illinois, by the writer, under the general direction of Professor Arthur N. Talbot, head of the Department of Theoretical and Applied Mechanics.

In addition to tension tests of zinc, tests of rolled zinc under punching and shearing were made. In the course of the investigation, several tests of cast zinc were made, including torsion and compression tests, and also cold bending tests of rolled zinc. The results of all these tests are recorded in this bulletin.

3. *Existing Data on the Strength of Rolled Zinc.*—The principal tests on the strength of rolled zinc have been made by Bauschinger, Martens, and Meyer. Bauschinger's tests* showed the effect of rapidity of loading on the properties of zinc. The influence of duration of test on the strength of cast zinc was slight, but with rolled zinc, rapidity of testing increased the tensile strength. In tests lasting 6 min., the average ultimate tensile strength was 29 100 lb. per sq. in.; in tests lasting 81 min., the average ultimate tensile strength was 23 300 lb. per sq. in.

TABLE 1.

TENSION TESTS OF ROLLED ZINC; EFFECT OF THICKNESS AND OF DIRECTION OF ROLLING (MARTENS).

Thickness of Plate—inches	Ultimate Tensile Strength lb. per sq. in.	
	With Grain (Parallel to Direction of Rolling)	Across Grain (Perpendicular to Direction of Rolling)
0.019—0.222	26 600—14 260	31 800—20 200

TABLE 2.

TENSION TESTS OF ROLLED ZINC; EFFECT OF TEMPERATURE OF TEST SPECIMEN (MARTENS).

Temperature of Specimen Degrees Fahr.	Ultimate Tensile Strength lb. per sq. in.	Elongation after Rupture per cent
66—81	20 600	12.4
176	12 500	29.4
248	8 960	59.4
302	5 790	101.5
338	7 960	17.1
392	6 120	7.2

Martens'† investigations of the strength of zinc dealt with the influence on strength of thickness of plate, of direction of rolling, and of temperature. His results are summarized in Table 1 and Table 2. From these results, it would seem that about 300° F. is the most favorable rolling temperature for zinc plates, as at that temperature the strength is low and the ductility a maximum. All Martens' tests were made on zinc refined from Silesian ores.

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*Mitteilungen aus dem mechanisches—technische Lab. der Technische Hochschule im Munchen, 1887, Heft 20, Seite 16.

†Mitteilungen aus mech—tech. Versuchstantalten in Berlin, Erganzungsheft IV.

An extensive series of tests of the strength of zinc plates was made by Dr. Oswald Meyer of Vienna*. The plates tested by him were all rolled at the zinc works of Cilli, in Austria. The plates contained from 0.021 to 1 04 per cent of lead, 0.03 to 0 912 per cent of cadmium, 0.02 to 0.03 per cent of iron, from a trace to 0.009 per cent of copper, and a trace of arsenic. Tests were made of (a) plates as received from the zinc works; (b) plates treated with nitric acid; and (c) plates subjected to heat treatment before testing. The results of Meyer's tension tests of plates as received from the zinc works, are shown in Table 3.

TABLE 3.

TENSION TESTS OF ROLLED ZINC (MEYER).

Tests of zinc as received from the rolling mill.
Thickness of plate tested varied from 0.044 in. to 0 051 in.

Item	With Grain	Across Grain	Average
Stress at first permanent set, lb. per sq. in	1 990	2 420	2 130
Stress at limit of proportionality of stress to deformation, lb. per sq. in	710	1 280	995
Yield point, lb. per sq. in	11 400	13 640	12 500
Ultimate, lb. per sq. in	30 400	36 800	33 600
Elongation, per cent	27.2	9.7	18.5
Reduction of area, per cent	43	17	30
Modulus of elasticity, lb. per sq. in	12 850 000	14 500 000	13 620 000

The following features of Meyer's tests are worthy of note: Specimens cut across grain (perpendicular to the direction of rolling) are somewhat stronger and stiffer than specimens cut with the grain (parallel to the direction of rolling).

The ductility of specimens cut with the grain is greater than that of specimens cut across the grain. Stresses at elastic limit and yield point are very low, the yield point is not sharply marked, and the flow of metal under high stress goes on for a long time.

The tests of zinc plates treated with acid show that a 10 min. immersion in 5 per cent nitric acid did not appreciably lessen the strength or the ductility of the zinc plate.

A series of tests was made by Dr. Meyer on zinc plates subjected to the following heat treatment before testing in tension:

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*Oesterreichische Zeitschrift fur Berg und Huttenwesen, Oct. 7 and 14, 1905.

The specimens were subjected for one hour to a temperature of 527° F., when upon testing they developed the properties indicated in Table 4 (average values).

The ultimate strength in tension and the ductility are decidedly lowered by this heat treatment. The "critical temperature" for zinc was found to be at about 300° F. and the heat transformation of the zinc took place very rapidly, one minute being seemingly sufficient to effect it. The tests of Martens and of Meyer point to the desirability of keeping the working temperature during the rolling of zinc within narrow limits. Meyer recommends 302° F. as a maximum rolling temperature.

Meyer also made tests to show the effect of alloying zinc with cadmium and with lead. He found that the addition of 0.2 per cent of cadmium improved the quality of zinc plate, but that the addition of 0.4 per cent of either cadmium or lead either produced no appreciable effect or injured the quality. The addition of both cadmium and lead lowered both the strength and the ductility of zinc plate.

The Matthiessen and Hegeler Zinc Company of La Salle, Illinois, report a series of tension tests of zinc plate made at their request in 1907. The thickness of the plate tested varied from 0.011 in. to 0.04 in., the average tensile strength for sixty specimens was 29 370 lb. per sq. in. The thinner plate showed slightly greater strength than the thicker plate.

II. SPECIMENS, TESTS, AND METHODS OF TESTING.

4. *Source of Supply of Zinc for Tests.*—The zinc tested at the University of Illinois came from the zinc works of the Matthiesson and Hegeler Zinc Company at La Salle, Illinois, and from the stock of a local hardware store. The zinc from the Matthiessen and Hegeler Company was smelted from ores from the Joplin, Missouri, district. Several specimens of cast zinc were furnished which were remelted spelter poured directly into moulds. Eighteen sheets of rolled zinc, each 18 in. by 20 in., were furnished, varying in thickness from 0.006 in. to 1.0 in. Three sheets of each thickness were furnished, and each of the three sheets was from a different heat. No special precautions as to heat treatment were taken either with the cast zinc or with the rolled zinc. As a check on the values obtained for the zinc plates furnished by the Matthiessen and Hegeler Zinc Co., tests were made on sheet zinc purchased in the local market (Champaign, Illinois).

TABLE 4.

Tension Tests of Heat Treated Zinc (Meyer).

Stress at first permanent set	2 480 lb. per sq. in.
Stress at limit of proportionality of stress to deformation	1 710 lb. per sq. in.
Yield point	11 100 lb. per sq. in.
Ultimate	17 500 lb. per sq. in.
Elongation	4.6 per cent
Reduction of area	5 per cent
Modulus of elasticity	15 950 000 lb. per sq. in.

5. *Test Specimens.*—Specimens of cast zinc were tested in tension in compression, and in torsion. Specimens of rolled zinc were tested in tension, in cold bending and in shear; the tests in shear included punching tests and tests in direct shearing. The specimens for tension tests of cast zinc were similar in form and dimension to the specimen shown in Fig. 1. The cast zinc was furnished in bars 1¼ in. square by 12 in. long and machined to the size shown in Fig. 1.

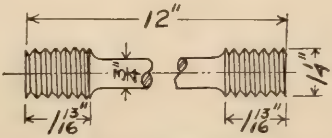

Fig. 1. Tension Specimen for Cast Zinc.

Fig. 2 shows the form and dimension for the specimens of cast zinc tested in torsion. The specimens of cast zinc tested in compression were circular cylinders 1 in. in diameter by 1½ in. long. They were machined all over.

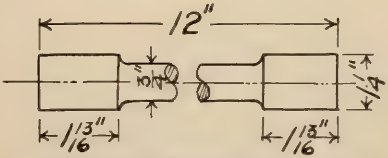

Fig. 2. Torsion Specimen for Cast Zinc.

Fig. 3 shows the size of the plates of rolled sheet zinc furnished by the Matthiessen and Hegeler Company, and also shows the plan followed in cutting test specimens from the plates. The specimens similarly located in different plates were stamped with the same number, the plate being designated by a letter. The specimens for tension tests of rolled zinc were similar in form and dimension to the specimen shown in Fig. 4. The strips from which they were machined out were sheared from an 18 in. by 20 in. plate in all cases except some of the plates 1.0 in. thick, in which it was found impossible to shear strips from the plate without seriously injuring it. In these thick plates, the strips from which the tension specimens were machined were cut from the plates by drilling. The tension specimens of rolled zinc were machined on a shaper to the shape shown in Fig. 4.

The punching and shearing tests for rolled zinc were made on the portion of the 18 in. by 20 in. plates remaining after the tension test pieces had been cut out. The holes punched in the punching tests were separated by a distance at least 1½ times the diameter of the punch used. The shearing tests were made on specimens cut from the portion of the plates left after the punching tests had been made. The shearing test specimens were cut 1 in. wide, and were sheared in double shear. Some specimens were sheared across the grain (perpendicular to the direction of rolling) and some with the grain (parallel to the direction of rolling).

Cold bending tests were made on small strips of zinc about 1 in. wide cut from any portion of the 18 in. by 20 in. plates remaining after the other tests had been made. For each plate tested in cold bending, one specimen was bent in a plane parallel to the direction of rolling (with the grain), and one specimen was bent in a plane perpendicular to the direction of rolling (across the grain).

The sheet of zinc bought in the local market differed in size from those furnished by the Matthiessen and Hegeler Company, but the general method of cutting specimens from it was the same as that just described.

6. *Testing Machines and Auxiliary Apparatus.*—Some of the tension tests of zinc were made on a Riehle 100 000-lb. testing machine, others on an Olsen 10 000-lb. testing machine, and still others on a Riehle 100 000-lb. testing machine fitted with an Olsen 1000-lb. spring balance for measuring load. The tension specimens for cast zinc were held in threaded-ended sockets; the tension specimens for rolled zinc were held by means of flat wedge grips. For measuring the elongation in the tension tests, the extensometer used in the routine testing of the Laboratory of Applied Mechanics for several years was used. The indicating pointer of this extensometer is operated by a drum round which is wrapped a fine insulated copper wire which by friction causes the rotation of the drum and the pointer over a dial as the specimen stretches. The extensometer read to $\frac{1}{10\,000}$ in. elongation.

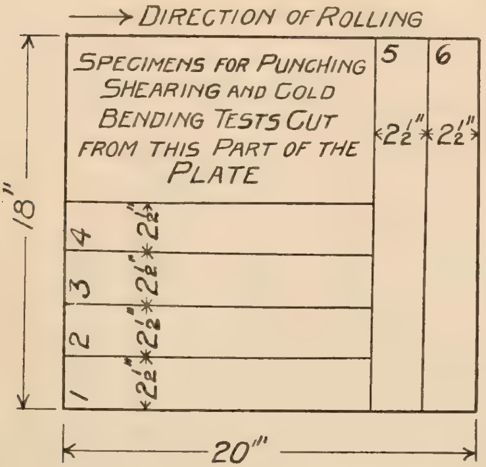
DIRECTION OF ROLLING
SPECIMENS FOR PUNCHING SHEARING AND COLD BENDING TESTS CUT FROM THIS PART OF THE PLATE
5
6
2½"
2½"
18"
4
3
2
1
2½"
2½"
2½"
2½"
20"

FIG. 3. ZINC PLATE SHOWING LOCATION OF SPECIMENS.

The torsion specimens of cast zinc were tested in a Riehle 10 000–lb. in. torsion testing machine of the pendulum type. The machine was hand operated during the tests, and the specimens held by self-centering and self-tightening toothed jaws. The angle of twist was measured by means of two dials and pointers similar to the dial and pointer on the extensometer used in the tension tests. The reading of each dial showed the twist of a section of the specimen with reference to the framework of the testing machine; the difference of the dial readings at the two sections gave the angle of twist of the specimen between the two sections at which the dials were attached.

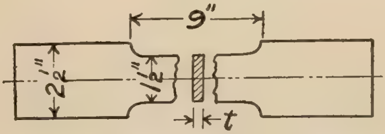

FIG. 4. TENSION SPECIMEN FOR ROLLED ZINC.

The compression specimens of cast zinc were tested in a 100 000-lb. Riehle testing machine. The ends of the specimens were carefully machined to a plane surface, and pressure was transmitted to the specimen through a spherical seated block. Compression was measured by means of a Ewing microscope compressometer reading to $\frac{1}{125\,000}$ in.

The punching tests of rolled zinc were made with hardened steel punches and dies mounted on the weighing table of a testing machine. The punches used were 0.505 in., 0.747 in., and 1.001 in. in diameter, respectively, and the corresponding dies were 0.61 in., 0.86 in., and 1.13 in. in diameter, respectively. The punches were flat faced. Most of the punching tests were made with a speed of punch so slow (0.1 in. per min.) that the weighing beam of the testing machine could be kept in balance by hand as the test progressed. Nearly all the punching tests were made on a Riehle 100 000-lb. testing machine fitted with an autographic apparatus which drew a diagram showing motion of punch as abscissas and load applied as ordinates. It was desired to run some punching tests at a higher speed than 0.1 in. per min., but as it was impossible to keep the weighing beam accurately balanced by hand in these tests, a steam engine indicator was attached to the testing machine weighing beam, so that the compression of the indicator spring measured the load on the punching tool*. Punching tests of very thin plates of zinc were made on an Olsen 10 000-lb. testing machine, and only the maximum load was recorded.

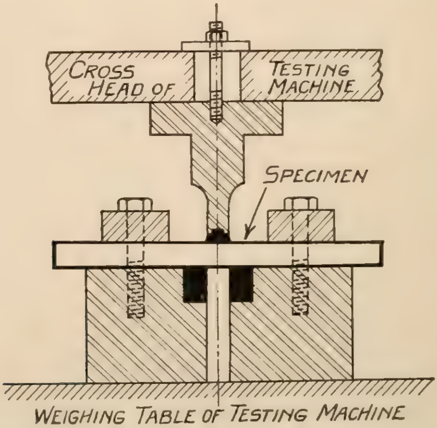
CROSS HEAD OF TESTING MACHINE
SPECIMEN
WEIGHING TABLE OF TESTING MACHINE

FIG. 5. APPARATUS FOR SHEARING TESTS.

Shearing tests of rolled zinc were made in a 100 000-lb. Riehle testing machine fitted with autographic recording apparatus. A hardened steel shearing tool cut the specimen in double shear. Fig. 5 shows the arrangement of the shearing tool. Most of the shearing tests were run at a low speed of tool (0.1 in. per min.) and the weighing beam of the testing machine was kept balanced by hand, while an autographic record was taken of the load and the motion of the shearing tool. Some shearing tests were made at a higher speed, and for these the steam engine indicator device for weighing loads was used as in the punching tests at high speeds.

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*For description of this apparatus see the Proceedings of the American Society for Testing Materials for 1908, p. 653.

7. *Data of Tension Tests.*—Table 5 gives the log of a sample tension test of zinc. The same general method was followed for all tension tests; both deformation under load and set after release of load were observed. During several of the tests, especially in the tests of cast zinc, a cracking noise was plainly audible under stresses as low as two-thirds of the ultimate. A set was detected in most tests after the removal of the first load applied, however low. After rupture the elongation over a gauge length originally measuring 8 in. and the reduction of area at the point of fracture were both measured when the rupture was inside the gauge length.

TABLE 5.

LOG OF SAMPLE TENSION TEST OF ROLLED ZINC.

Specimen H. 3. Dimension of Cross-section, 1.486 in. x 0.612 in.
Elongation after rupture 9.5% in 8 in.
Reduced cross-section 1.439 in. x 0.544 in.
Specimen tested with the grain.

Load lb.	Extensometer in.	Load lb.	Extensometer in.
1 000	0	8 000	0.0091
3 000	0.0015	1 000	0.0054
1 000	0.0006	8 750	0.0120
5 000	0.0031	1 000	0.0076
1 000	0.0011	9 500	0.0155
7 000	0.0066	1 000	0.0104
1 000	0.0035	19 530	rupture

In the tests of cast zinc and of the thinner specimens of rolled zinc the failure was, in general, sudden. In the very thin specimens of sheet zinc the failure occurred by tearing across, and in a few cases its course could be followed by the eye. Fig. 6 and 7 show typical stress-elongation diagrams of tension tests. Table 6 shows the summarized results of tension tests of zinc. There was considerable variation in strength shown by individual specimens of cast zinc, the ultimate strength ranging from 6 050 lb. per sq. in. to 12 220 lb. per sq. in. There was little variation in strength shown by individual specimens of rolled zinc. The extreme values were, in general, within 10 per cent of the mean. No

TABLE 6.

TENSION TESTS OF ZINC.

The values given are the average results for the number of specimens noted in the second column.

Thickness of Rolled Plate in.	Number of Specimens	Stress at First Noticeable Set lb. per sq. in.	Stress at Limit of Proportionality lb. per sq. in.	Stress at Ultimate lb. per sq. in.	Modulus of Elasticity lb. per sq. in.	Elongation in 8 in. per cent	Reduction of Area per cent	Speed of Pulling in. per min.	Specimens Tested with or across Grain
Cast	6	1600	2100	9 060	11 025 000	slight	slight	hand	——
1	5	0+*	0+	22 200	——	7.56	8.72	0.3	With grain
1	4	900	2900	21 340	10 450 000	4.85	6.58	0.5	With grain
1	3	900	4600	23 050	10 370 000	0.31	5.37	0.5	Across grain
0.6	12	0+	4200	21 490	11 033 000	16.63	22.25	0.5	With grain
0.6	5	0+	4000	23 550	10 780 000	3.33	5.80	0.5	Across grain
0.25	12	0+	5150	23 770	12 767 000	11.90	19.17	0.5	With grain
0.25	6	0+	5000	22 260	10 967 000	0.27	5.07	0.5	Across grain
0.10	6	0+	3600	22 280	10 650 000	18.75	——	0.5	With grain
0.10	6	——	——	24 870	——	21 00	——	3.25	With grain
0.10	6	0+	5500	33 620	13 700 000	14.3	——	0.5	Across grain
0.018	6	0+	5800	25 490	10 533 000	——	——	0.5	With grain
0.018	9	——	——	29 810	——	——	——	1.37	Across grain
0.018	3	——	——	23 840	——	——	——	0.2	With grain
0.018	6	——	——	34 960	——	——	——	0.5	Across grain
0.006	6	——	——	24 110	——	——	——	0.2	With grain
0.006	6	0+	5100	26 480	14 800 000	6.5	——	0.5	With grain
0.006	3	——	——	33 830	——	——	——	0.5	Across grain
0.006	3	——	——	38 440	——	——	——	2.5	Across grain
0.018†	3	——	——	24 200	——	——	——	0.5	With grain
0.018†	2	——	——	35 150	——	——	——	0.5	Across grain

*When the stress is given as 0+, a value below the lowest reading taken is denoted.

†Specimens from zinc bought in the local market; all specimens not specially designated were cut from zinc furnished by the Matthiessen and Hegeler Zinc Company.

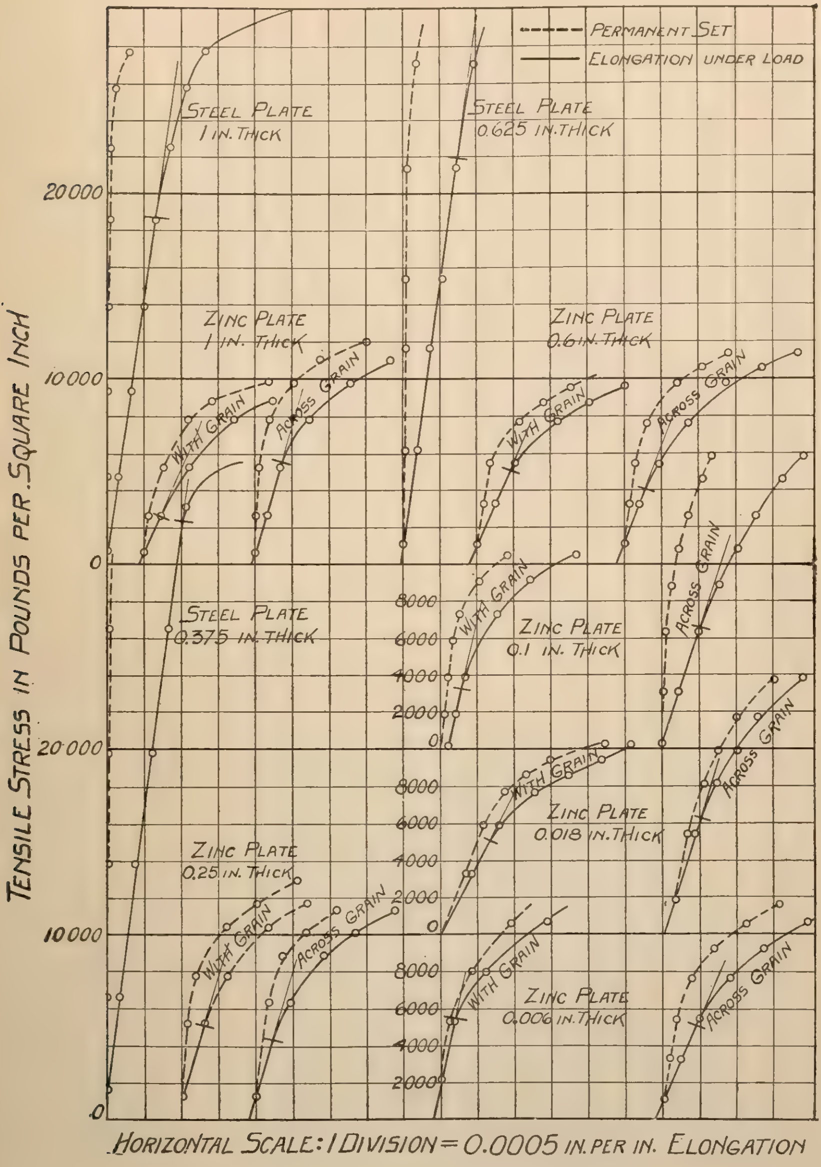

FIG. 6. TENSION TEST DIAGRAMS FOR ROLLED ZINC.

well-defined yield point could be determined in any tension test.

Fig. 8 shows two characteristic fractures of cast zinc in tension. The specimen at the left of the cut shows a much coarser grain than the one on the right; it also showed much lower tensile strength. Fig. 9 shows characteristic fractures of rolled zinc in tension. The specimen at the left is from a very thin plate, and it failed by tearing across. The specimen in the center is from a plate 0.25 in. thick, and it failed with very little elongation. The specimen at the right is from a plate 0.6 in. thick, and it showed great elongation. The necking down of the ductile specimen can be seen in Fig. 9.

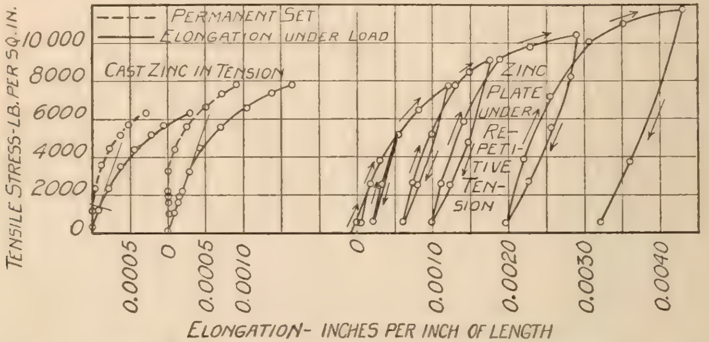

FIG. 7. TENSION TEST DIAGRAMS FOR ZINC.

8. *Data of Punching and Shearing Tests*.—The data for the punching and the shearing tests were all autographic except for punching tests in very thin plates. Fig. 10 shows typical diagrams for punching tests, and Fig. 11 shows typical diagrams for shearing tests of zinc plate. In Fig. 11 are also shown diagrams of shearing tests of steel plate.

The results of the punching tests are summarized in Table 7 and the shearing tests in Table 8. The variation of the extreme values for ultimate strength of individual specimens from the average values reported in Tables 7 and 8 was, in general, not greater than 10 per cent. The variation in the amount of energy required was somewhat greater. As one of the principal items of information desired was a comparison between zinc and steel as to maximum stress developed and amount of energy required in punching and shearing, punching and shearing tests were made on mild steel plates and the results are summarized in Tables 9 and 10.

Fig. 12 shows the appearance at several stages of the punching process of the "wad" of zinc as it is being pushed out ahead

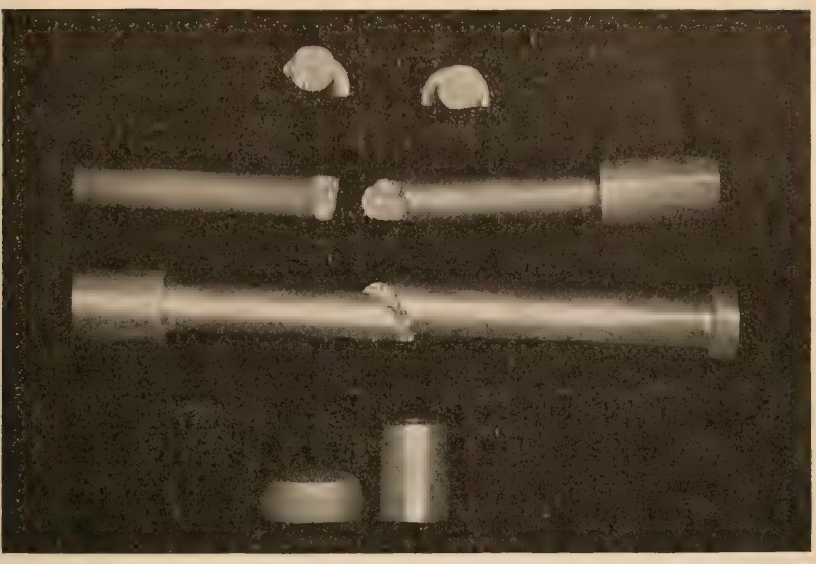

FIG. 8. SPECIMENS OF CAST ZINC AFTER TESTING.

of the punching tool. Fig. 12 also shows a shearing specimen which is about to fail. The distortion of the tool marks, originally straight, shows in a general way the distortion of the fibers of the specimen.

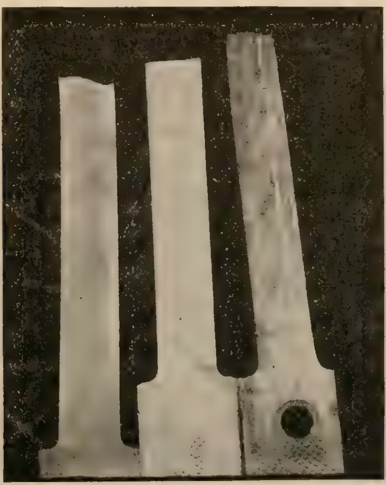

FIG. 9. SPECIMEN OF ROLLED ZINC AFTER TESTING IN TENSION.

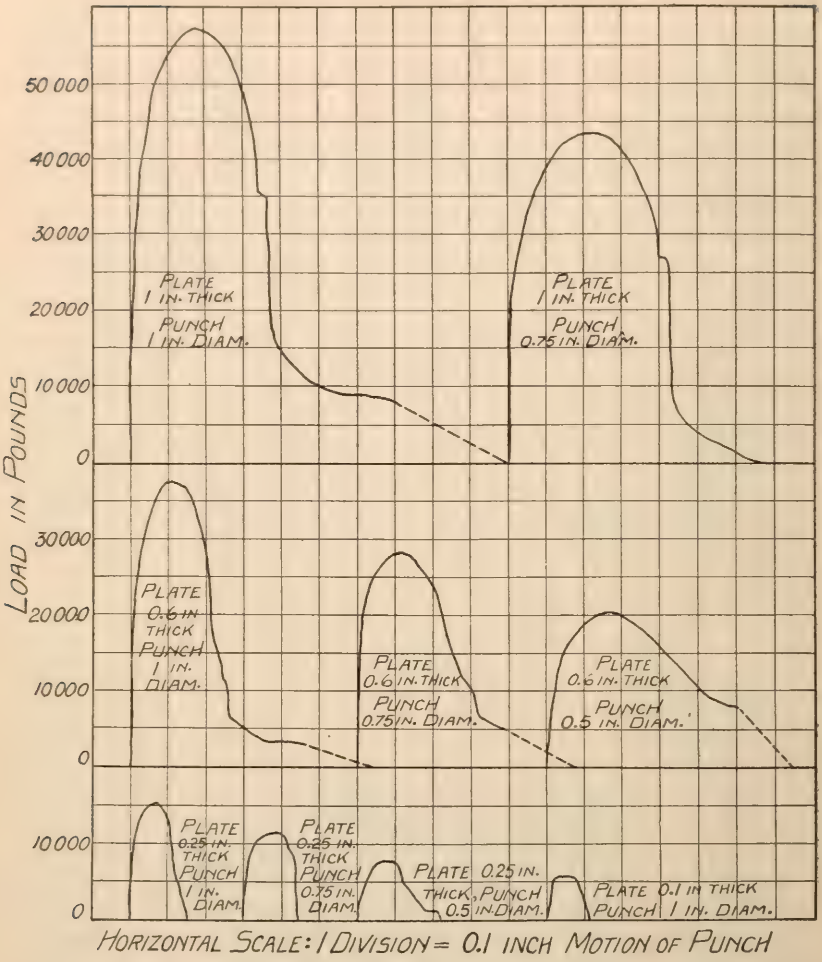

FIG. 10. PUNCHING TEST DIAGRAMS FOR ROLLED ZINC.

9. *Miscellaneous Tests.*—Torsion tests were made on six test pieces of cast zinc. The results of these tests are summarized in Table 11, and a typical stress-deformation diagram for torsion of cast zinc is shown in Fig. 13. In Fig. 8 are shown torsion test specimens after rupture. Attention is called to the character of the fracture, and its similarity in form to that of cast iron under torsion.

Compression tests were made on four short cylinders of cast zinc. The results of these tests are given in Table 12, and a typical stress-compression diagram for cast zinc is shown in Fig.

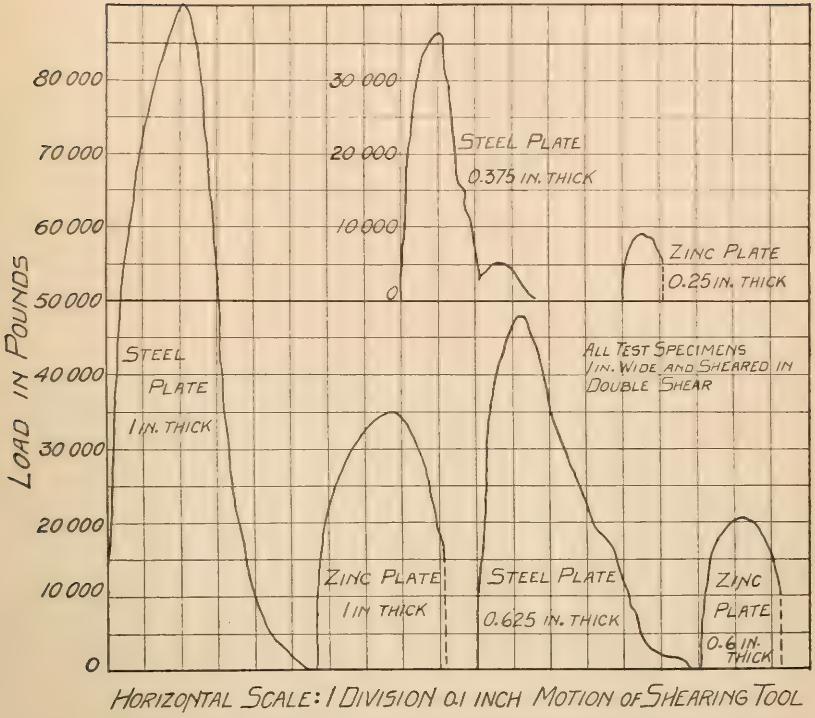

Fig. 11. Shearing Test Diagrams for Rolled Zinc and for Steel.

14. Fig. 14 also shows a stress-compression diagram for zinc under steadily increasing repetitive loading. Both in the repetitive loading test in compression and in tension (see Fig. 7) there was an appreciable loss of energy during the release and the reapplication of a load. This loss of energy, "mechanical hysteresis" as

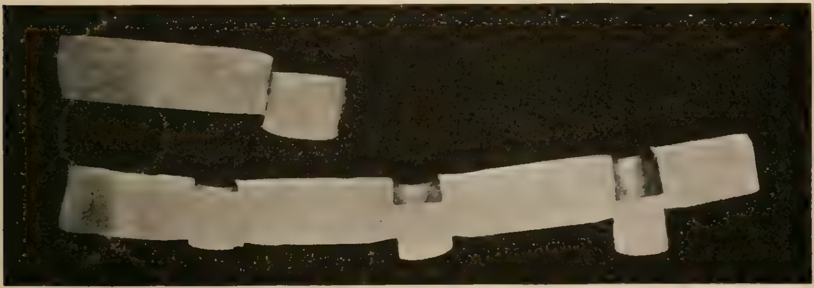

Fig. 12. Punching and Shearing Test Specimens after Testing.

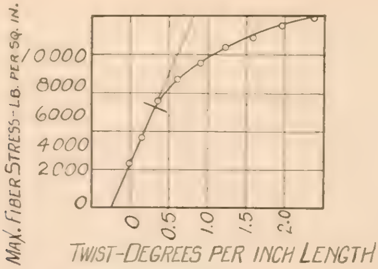

FIG. 13. TORSION TEST DIAGRAM FOR CAST ZINC.

it is called, is shown by loops in the diagrams of tests under repetitive load.

The behavior of cast zinc under compression is worthy of note. Judged by fractures in tension and in torsion tests, cast zinc is a brittle material, and under compression a shattering failure might be expected—such a failure as cast iron exhibits in compression. What actually happened to the compression test pieces of cast zinc was a gradual flattening out, such as occurs with soft steel. No maximum load could be determined. Cast zinc is evidently a plastic but not a ductile metal. Fig. 8 shows a cylinder of cast zinc before testing, and beside it another originally of the same size, after compression under 100 000 lb.

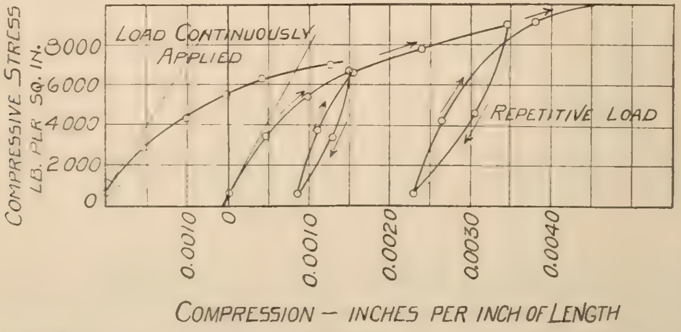

FIG. 14. COMPRESSION TEST DIAGRAMS FOR CAST ZINC.

Cold bending tests were made on specimens from all rolled plates except those 1 in. thick and those 0.6 in. thick. One test was made on specimens from a plate 0.6 in. thick. From each plate tested one specimen was bent in a plane parallel to the direction of rolling and one specimen bent in a plane perpendicular to the direction of rolling. Table 8 shows the results of the cold bending tests.

TABLE 7.

PUNCHING TESTS OF ZINC.

The values given are the average results for the number of specimens noted in the second column.

Nominal Thickness of Plate in.	Number of Specimens Tested	Diameter of Punch in.	Speed of Punching in. per min.	Ultimate Shearing Stress Developed lb. per sq in.	Energy Required to Punch in lb. per sq. in. per in. thickness
1.00	9	0.75	0.10	18 750	7 260
1 00	9	1.00	0.10	19 060	6 650
0.60	6	0.50	0.10	19 740	10 590
0.60	6	0.75	0.10	19 350	8 630
0.60	6	1.00	0.10	18 990	6 400
0.25	6	0.50	0.10	20 000	10 550
0.25	6	0.75	0.10	19 700	8 740
0.25	6	1.00	0.10	19 390	8 030
0.25	6	1.00	0.50	19 400	8 350
0.25	6	1.00	1.60	20 850	8 380
0.10	6	1.00	0.10	18 230	5 650
				Av. 19 400	8 110
0.018*	9	1.00	0.10	12 930	——
0.018*	6	1.00	0.10	13 380†	——
0.018*	3	1.00	0.10	10 880	——

*In the very thin plates, shearing took place unevenly around the circumference of the punching tool.

† Zinc bought in the local market.

TABLE 8.

SHEARING TESTS OF ZINC.

The values given are the average results for the number of specimens noted in the second column.

Nominal Thickness of Plate in.	Number of Specimens Tested	Speed of Shearing Tool in. per min.	Ultimate Shearing Stress Developed lb per sq. in.	Energy Required to Shear in. lb per sq. in. per in. thickness	Shearing with Grain or across Grain
1.00	6	0.10	16 700	4 850	Across grain
1.00	3	0.10	16 580	4 510	With grain
1.00	6	0.50	17 140	4 130	Across grain
1.00	6	1.60	17 770	4 330	Across grain
0.60	6	0.10	16 580	5 100	Across grain
0.60	3	0.10	17 380	4 620	With grain
0.60	6	0.50	15 480	4 640	Across grain
0.60	6	1.60	18 100	4 410	Across grain
0.25	6	0.10	18 860	6 690	Across grain
0.25	3	0.10	18 170	4 620	With grain
0.25	6	0.50	15 480	4 640	Across grain
0.25	6	1.60	17 040	5 730	Across grain
			Av. 17 100	4 850	

III. Results and Conclusions.

10. *Tensile Strength of Zinc.*—An examination of the results of tension tests shows that zinc, either cast or rolled, is imperfectly elastic under very low stresses. The results of Meyer's tests are in agreement with the results of the Illinois tests as regards this general conclusion, though in the Illinois tests permanent set was detected at lower stresses than in Meyer's tests. Whether the elastic limit be defined as the lowest stress under which a material is given a permanent set, or as the lowest stress at which Hooke's law (proportionality of stress to deformation) is found to be inexact, the elastic limit of zinc is very low and very poorly defined. It is doubtful if the elastic limit as determined either by Meyer or in the Illinois tests has any special physical significance. Though Meyer reported a yield point of zinc in the tables of results of his tests, in the accompanying discussion he stated that the yield point was poorly defined. In the Illinois tests no well-defined yield point could be detected, and none was reported.

The ultimate tensile strength of cast zinc depends on the temperature of pouring, and other factors, and varies between wide limits. Thin plates of rolled zinc are relatively stronger under tension than thick plates. From the results of the Illinois tests for plates under 0.05 in. thick, 24 000 lb. per sq. in. would seem to be about the value to be used for the ultimate tensile strength of rolled zinc. For plates over 0.05 in. thick, 21 000 lb. per sq. in. would seem a reasonable value to use. Meyer's tests and those of the Matthiessen and Hegeler Zinc Company show slightly higher values. Rolled zinc is somewhat stronger in tension across the grain than in tension with the grain. A higher tensile strength of rolled zinc was obtained by increasing the rapidity of application of load. Rapidity of testing may, in part, account for the fact that higher values of tensile strength were found by Meyer and by Matthiessen and Hegeler than were found at Illinois. Neither of the first two reports the speed of testing. The speed used in the Illinois tests was lower than is sometimes used in commercial testing.

From the results of the various tension tests of zinc herein quoted, it would seem that the modulus of elasticity of zinc is about 11 500 000 lb. per sq. in.

11. *Shearing Strength of Rolled Zinc.*—The values of shearing strength of rolled zinc reported in this bulletin were determined with the purpose of throwing some light on the problem of what

TABLE 9.

PUNCHING TESTS OF STEEL PLATE.

For purposes of comparison with punching tests of zinc plates.

The values given are the average results for the number of specimens noted in the second column.

Nominal Thickness of Plate in.	Number of Specimens Tested	Diameter of Punch in.	Speed of Punching in. per min.	Ultimate Shearing Stress Developed lb. per sq. in.	Energy Required to Punch in. lb. per sq. in. per in. thickness
0.375	2	0.75	0.10	50 000	30 350
0.375	2	1.00	0.10	50 080	19 880
				Av. 50 040	25 200

TABLE 10.

SHEARING TESTS OF STEEL PLATE.

For purposes of comparison with shearing tests of zinc plates.

The values given are the average results for the number of specimens noted in the second column.

Nominal Thickness of Plate in.	Number of Specimens Tested	Speed of Shearing Tool in. per min.	Ultimate Shearing Stress Developed lb. per sq. in.	Energy Required to Shear in. lb. per sq. in per in. thickness	Shearing with Grain or across Grain
1.000	2	0.10	44 700	11 580	Across grain
0.625	2	0.10	37 200	17 750	Across grain
0.375	3	0.10	46 970	18 850	Across grain
0.375	3	0.50	43 000	16 740	Across grain
0.375	2	1.60	44 500	19 630	Across grain
			Av. 43 270	16 910	

TABLE 11.

TORSION TESTS OF CAST ZINC.

Number of specimens tested..	6
Maximum fiber stress at limit of proportionality of stress to angle of twist....	5 450 lb. per sq. in.
Computed maximum fiber stress at rupture..........	15 260 lb. per sq in.
Modulus of elasticity in shear (Torsion)...............	4 570 000 lb. per sq. in.

TABLE 12.

COMPRESSION TESTS OF CAST ZINC.

Number of specimens tested..	4
Fiber stress at limit of proportionality of stress to compression	1 620 lb. per sq in.
Modulus of elasticity..........	6 900 000 lb. per sq. in.

sizes of punches and shears should be used in working with zinc plates. Two factors which are important in their influence on the design of punches and shears are the maximum force to be exerted during the process of punching or shearing and the energy required to complete the punching or shearing action. The maximum force to be exerted determines the strength of frame, ram, gearing, and other parts of the punch or shear; the energy required determines, in large measure, the weight of the flywheel, size of belt, and power required for power-driven punches and shears, or the power and amount of water or air required for hydraulic or pneumatic punches or shears. The maximum force to be exerted during the punching or shearing action is in general proportional to the area actually sheared under the action of punching or shearing tool; the mean force during the action is approximately proportional to the maximum force and consequently to the area under the action of the punching or shearing tool. (This may be seen from the general similarity of shape of the punching and shearing diagrams for different thicknesses of plate; see Fig. 10 and 11). The energy required to punch or shear any plate will then be approximately proportional to the area to be sheared (by punching or shearing tool) multiplied by the distance traveled by the tool during the punching or shearing action, i. e., by the thickness of the plate. The significant features of the punching or shearing tests of rolled zinc were, then, the maximum shearing stress developed measured in pounds per square inch, and the energy required for punching or shearing measured in inch-pounds per square inch of surface sheared per inch thickness. An examination of the results of the punching and shearing tests of zinc plate shows an average value of shearing stress developed of 19 400 lb. per sq. in. for the punching tests, and of 17 100 lb. per sq. in. for the shearing tests. The average value of the energy required was 8 110 in. lb. per sq. in. per in. thickness for the punching tests, and 4 850 in. lb. per sq. in. per in. thickness for the shearing tests. Evidently the frictional resistance of the metal pushed out (the "wad") is greater in punching than in shearing, as is shown by the slightly greater stress developed, and by the markedly greater energy required.

An examination of the results of the punching tests shows that the larger the punch for any given thickness of plate the less the unit-energy required to punch the plate. In both punching and shearing tests, the maximum stress developed was slightly increased under increased speed of punching or shearing tool.

TABLE 13.

COLD BENDING TESTS OF ROLLED ZINC.

Nominal Thickness of Plate in.	Specimen Bent with Grain or across Grain	Action under Cold Bending
0.006	Across Grain	Bent double and hammered flat without cracking.
0.006	With Grain	Bent double and hammered flat without cracking.
0.018	Across Grain	Bent double and hammered flat without cracking.
0.018	With Grain	Bent double and hammered flat without cracking.
0.018*	Across Grain	Bent double and hammered flat without cracking.
0.018*	With Grain	Bent double and hammered flat without cracking.
0.100	Across Grain	Two specimens bent double and hammered flat without cracking; one specimen cracked when hammered flat.
0.100	With Grain	All specimens bent double and hammered flat without cracking.
0.250	Across Grain	Specimens cracked when bent through 90–120 degrees.
0.250	With Grain	One specimen bent double and hammered flat without cracking; two specimens cracked after bending through 180 degrees.
0.600	Across Grain	One specimen tested, broke short off when bent through about 30 degrees.
0.600	With Grain	One specimen tested, cracked after bending through 180 degrees.

* Zinc bought in local market.

Both punching and shearing tests were made with flat-faced tools, the object being to bring stress as uniformly as possible on all parts of the area under shear. By the use of beveled punching and shearing tools, the maximum force resisting shear would have been reduced.

An examination of the results of the punching and shearing tests of mild steel plate tested for purposes of comparison with rolled zinc shows the following average values: Shearing stress developed, 50 040 lb. per sq. in. for the punching tests, 43 270 lb. per sq. in. for the shearing tests. Energy required 25 200 in. lb. per sq. in. per in. thickness for the punching tests, and 16 910 in. lb. per sq. in. per in. thickness for the shearing tests*. In punching or shearing zinc plates, about 40 per cent as high a stress is developed as is developed in punching or shearing mild steel plates of the same size, and about 30 per cent as much energy is required.

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*Other punching and shearing tests of mild steel give results not widely differing from these. See article by H. V. Loss in the American Engineer and Railroad Journal for March 1893 and results in Kent's Mechanical Engineers' Pocket Book.

12. *Ductility and Plasticity of Zinc.*—For the comparison of the ductility of different metals there is, unfortunately, no well-defined quantitative standard. In the series of tests described in this bulletin the elongation and the reduction of area after rupture in tension and the results of cold bending tests were all used to throw light on the ductility of the zinc tested. From the results of the tests, it is evident that zinc is much less ductile than wrought-iron or mild steel, and that it is less ductile across the grain than with the grain. For zinc plate which is to be stamped or bent into shape (for example in the making of zinc elements for dry batteries), a severe cold-bending test would seem to be of considerable value in determining the acceptability of a shipment of zinc plate.

The researches of Martens and of Meyer on the effect of heat treatment of zinc on its strength and ductility would indicate the desirability of measuring and of controlling the temperature in the rolling process and show the danger of rolling at too high a temperature. While the ductility of zinc is low as compared with that of steel, from the low and poorly defined elastic limit, from the loss of energy in "mechanical hysteresis" and from the behavior of compression test pieces it is evident that the zinc possesses a relatively high degree of plasticity.

13. *Summary.*—The following summary is given:

1. Zinc either rolled or cast has no well-defined yield point and its elastic limit is very low. Zinc possesses a relatively high degree of plasticity.

2. The ultimate tensile strength of thin rolled zinc plate (not more than 0.05 in. thick) is about 24 000 lb. per sq. in.

3. The modulus of elasticity of zinc in tension is about 11 500 000 lb. per sq. in.

4. The stress per square inch of area sheared developed in punching or shearing rolled zinc plates is about 40 per cent of the stress developed in punching or shearing mild steel plates.

5. The energy per square inch of area sheared per inch thickness of plate required to punch or shear rolled zinc plates is about 30 per cent of the energy required to punch or shear mild steel plates.

6. The ductility of rolled zinc is much less than that of mild steel, and the ductility of zinc plate with the grain is less than the ductility across the grain.

ANNOUNCEMENT CONCERNING A MODIFICATION IN THE RULES GOVERNING THE DISTRIBUTION OF BULLETINS

The Board of Trustees of the University of Illinois voted, December 3, 1910, that a price should be affixed to certain University publications, among them being the Bulletin of the Engineering Experiment Station.

This action, so far as it concerns the bulletins of the Engineering Experiment Station, has for its purpose a threefold object:

(1) To provide a greater degree of control in the distribution of bulletins.

(2) To make possible the establishment and maintenance of a trade circulation through the regular publishing houses.

(3) To regulate the distribution of the reserve or "out-of-print" supply.

IT IS NOT INTENDED THAT THIS ACTION SHALL OPERATE IN ANY WAY TO ABRIDGE THE PRIVILEGES OF THOSE WHO HAVE HITHERTO RECEIVED THE BULLETINS OF THE STATION GRATUITOUSLY, OR TO PREVENT REASONABLE EXTENSIONS OF THE EXISTING MAILING LISTS.

Practice under the new procedure will be as follows:

(1) All bulletins, hereafter issued, will have a price printed upon the cover, together with the name of the London sales agent.

(2) EACH BULLETIN, HOWEVER, WILL STILL BE SUBJECT TO A FREE INITIAL DISTRIBUTION, AS HERETOFORE, ON THE BASIS OF EXISTING MAILING LISTS.

It will also be placed on sale with authorized agencies, both in this country and abroad.

(3) THERE WILL BE A LIMITED NUMBER OF COPIES AVAILABLE FOR FREE DISTRIBUTION, UPON REQUEST, AFTER THE INITIAL DISTRIBUTION.

(4) As the supply of each bulletin approaches exhaustion, it will be placed upon a reserve or "out-of print" list. THE EFFECT OF THIS ACTION WILL BE TO WITHDRAW SUCH BULLETINS FROM FREE DISTRIBUTION. Bulletins withdrawn from free distribution will be available to any applicant upon payment of the assigned price.

(5) A discount from the published price will be allowed to news agents regularly handling the bulletins.

W. F. M. GOSS
Director.

PUBLICATIONS OF THE ENGINEERING EXPERIMENT STATION

ALL BULLETINS, THE TITLES OF WHICH ARE NOT STARRED, WILL BE SENT FREE UPON APPLICATION.

******Bulletin No. 1.* Tests of Reinforced Concrete Beams, by Arthur N. Talbot. 1904 *None available.*

**Circular No. 1.* High-Speed Tool Steels, by L. P. Breckenridge. 1905. *None available*

**Bulletin No. 2.* Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks. 1905. *None available.*

**Circular No. 2.* Drainage of Earth Roads, by Ira O. Baker. 1906. *None available.*

**Circular No. 3.* Fuel Tests with Illinois Coal. (Compiled from tests made by the Technologic Branch of the U. S. G. S., at the St. Louis, Mo., Fuel Testing Plant, 1904-1907.) by L. P. Breckenridge and Paul Diserens. 1909. *Thirty cents.*

**Bulletin No. 3.* The Engineering Experiment Station of the University of Illinois, by L. P. Breckenridge. 1906. *None available.*

**Bulletin No. 4.* Tests of Reinforced Concrete Beams, Series of 1905, by Arthur N. Talbot. 1906. *Forty-five cents.*

**Bulletin No. 5.* Resistance of Tubes to Collapse, by Albert P. Carman. 1906. *Fifteen cents.*

**Bulletin No. 6.* Holding Power of Railroad Spikes, by Roy I. Webber. 1906. *Thirty-five cents.*

**Bulletin No. 7.* Fuel Tests with Illinois Coals, by L. P. Breckenridge, S. W. Parr and Henry B. Dirks. 1906. *Thirty-five cents.*

**Bulletin No. 8.* Tests of Concrete: I. Shear; II. Bond, by Arthur N. Talbot. 1906 *None available.*

**Bulletin No. 9.* An Extension of the Dewey Decimal System of Classification Applied to the Engineering Industries, by L. P. Breckenridge and G. A. Goodenough. 1906. *None available.*

**Bulletin No. 10.* Tests of Concrete and Reinforced Concrete Columns, Series of 1906, by Arthur N. Talbot. 1907. *None available.*

**Bulletin No. 11.* The Effect of Scale on the Transmission of Heat through Locomotive Boiler Tubes, by Edward C. Schmidt and John M. Snodgrass. 1907. *Fifteen cents.*

**Bulletin No. 12.* Tests of Reinforced Concrete T-beams, Series of 1906, by Arthur N. Talbot. 1907. *None available.*

**Bulletin No. 13.* An Extension of the Dewey Decimal System of Classification Applied to Architecture and Building, by N. Clifford Ricker. 1907. *Fifty cents.*

**Bulletin No 14.* Tests of Reinforced Concrete Beams, Series of 1906, by Arthur N. Talbot. 1907. *None available.*

**Bulletin No. 15.* How to Burn Illinois Coal without Smoke, by L. P. Breckenridge. 1908 *Twenty-five cents.*

**Bulletin No. 16.* A Study of Roof Trusses, by N. Clifford Ricker. 1908. *Fifteen cents.*

**Bulletin No. 17* The Weathering of Coal, by S. W. Parr, N. D. Hamilton, and W. F. Wheeler. 1908. *Twenty cents.*

**Bulletin No. 18.* The Strength of Chain Links, by G. A. Goodenough and L. E. Moore. 1908. *Forty cents.*

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